METAL MINING

1940

A. I. M. E.

Rec'd
12/2/40.

Wm R. Chedsey,
Rolla,
Mo.

TN
1
A5
vol. 141
N/C

TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS

(INCORPORATED)

and Petroleum

Volume 141

METAL MINING
1940

Sponsored by the
ROCKY MOUNTAIN FUND

PAPERS AND DISCUSSIONS PRESENTED AT MEETINGS HELD AT BIRMINGHAM, APRIL 7–9, 1937; VANCOUVER, SEPT. 13–20, 1937; NEW YORK, FEB. 14–17, 1938; FEB. 13–16, 1939; FEB. 12–15, 1940; AND AT TUCSON, NOV. 1–5, 1938.

PUBLISHED BY THE INSTITUTE
AT THE OFFICE OF THE SECRETARY
29 WEST 39TH STREET
NEW YORK, N. Y.

Copyright, 1940, by the
American Institute of Mining and Metallurgical Engineers
(Incorporated)

PRINTED IN THE UNITED STATES OF AMERICA

THE MAPLE PRESS COMPANY, YORK, PA.

FOREWORD

For the first time the Institute publishes a volume of TRANSACTIONS confined exclusively to metal mining. Of the twenty-three papers, seventeen deal with engineering and technologic phases of metal-mine operation and six concern health and safety in mines. It probably is true that progress in the arts of developing ore bodies, and of breaking, loading and transporting ore is, at the moment, less rapid than progress in some of the other fields covered by the Institute. For example, the adequate recording of advances in methods of "mining" of petroleum and in the technique of alloying metals involves the publication of more articles than are required for metal mining. Nevertheless, there have been important improvements in the equipment and methods used in metal mining and many of the articles in this volume are especially significant in registering these advances.

This volume, like Volume 112 on Milling Methods, published last year, is sponsored by the Rocky Mountain Fund through a generous appropriation recommended by the Committee, composed of Henry Krumb, *Chairman*, Harvey S. Mudd and George Otis Smith. J. Murray Riddell was chairman of the Committee on Mining Methods in 1937–1939 and most of the material in this volume was accumulated during that period. Guy N. Bjorge is chairman this year. Oliver Schaus headed the Committee on Health and Safety in Mines during the three-year period, 1937 to 1939, being succeeded this year by J. J. Carrigan. To each of these chairmen, to the members of their committees, and to the authors of the respective articles, the industry, the profession, and the Institute are indebted for this excellent volume.

A. B. PARSONS, *Secretary*.

NEW YORK, N. Y.
October 1, 1940.

A.I.M.E. OFFICERS AND DIRECTORS

For the year ending February, 1941

President and Director
H. G. Moulton, New York, N. Y.

Past Presidents and Directors
D. C. Jackling, San Francisco, Calif.
Donald B. Gillies, Cleveland, Ohio

Treasurer and Director
Karl Eilers, Sea Cliff, N. Y.

Vice-presidents and Directors
Paul D. Merica, New York, N. Y.
Wilfred Sykes, Chicago, Ill.
William B. Heroy, Houston, Texas
Henry Krumb, New York, N. Y.
Erle V. Daveler, New York, N. Y.
W. M. Peirce, Palmerton, Pa.

Directors
John M. Boutwell, Salt Lake City, Utah
Holcombe J. Brown, Boston, Mass.
Charles Camsell, Ottawa, Ont., Canada
Chester A. Fulton, Baltimore, Md.
H. T. Hamilton, New York, N. Y.
A. B. Jessup, Waverly, Pa.
W. E. McCourt, St. Louis, Mo.
James T. MacKenzie, Birmingham, Ala.
Harvey S. Mudd, Los Angeles, Calif.
Le Roy Salsich, Duluth, Minn.
Francis A. Thomson, Butte, Mont.
H. Y. Walker, New York, N. Y.
F. A. Wardlaw, Jr., Inspiration, Ariz.
Henry D. Wilde, Houston, Texas
F. E. Wormser, New York, N. Y.
William Wraith, New York, N. Y.
L. E. Young, Pittsburgh, Pa.

Secretary
A. B. Parsons, New York, N. Y.

Division Chairmen—Acting as Advisers to the Board
Edmund M. Wise (Institute of Metals), Bayonne, N. J.
T. V. Moore (Petroleum), Houston, Texas
Frank T. Sisco (Iron and Steel), New York, N. Y.
Charles E. Lawall (Coal), Morgantown, W. Va.
W. R. Chedsey (Education), Rolla, Mo.
W. M. Weigel (Industrial Minerals), St. Louis, Mo.

Staff in New York

Assistant Secretaries
Edward H. Robie
Louis Jordan
Chester Naramore

Assistant Treasurer
H. A. Maloney

Assistant to the Secretary
E. J. Kennedy, Jr.

Business Manager
"Mining and Metallurgy"
Wheeler Spackman

CONTENTS

PAGE
Foreword. By A. B. PARSONS 3
A.I.M.E. Officers and Directors. 4
Committees on Mining Methods and on Health and Safety in Mines. 7

PAPERS

Recent Improvements in Mining Practice on the Mesabi Range. By ARTHUR E. ANDERSON, J. MURRAY RIDDELL and GROVER J. HOLT. (T.P. 968, with discussion). 9
A New Method of Top Slicing at Kipushi, Katanga, Belgian Congo. By G. VAN ESBROECK and M. VAN WEYENBERGH. (T.P. 1078, with discussion). . . . 27
Preliminary Stripping of the Morenci Open Pit, Arizona. By WALTER C. LAWSON. (T.P. 980, with discussion) 47
Recent Improvements in the Mining Practice of the Tri-State District. By C. W. NICOLSON. (T.P. 905, with discussion) 62
Ymir Yankee Girl Gold Mines Ltd. By LAWRENCE B. WRIGHT and LESTER G. MORRELL. (T.P. 937). 81
Prospecting, Mining and Washing the Brown Iron Ores of Alabama. By CHARLES MORGAN. (T.P. 860). 107
Sublevel Caving, Large-pillar Method, at the Montreal Mine. By R. A. BOWEN. (T.P. 886, with discussion). 119
Shaft Sinking on the Gogebic Iron Range. By W. A. KNOLL and J. C. SULLIVAN. (T.P. 887, with discussion). 128
Sand Filling at the Homestake Mine. By A. J. M. ROSS. (T.P. 1075, with discussion). 146
Progress of "Big Blasting" at Climax. By F. S. MCNICHOLAS. (T.P. 936, with discussion). 165
Report of Committee on Classification of Surface Mining Methods. 179
Controlling Subsidence of a Large Inverted Cone of Barren Rock Lying above the Ore Body, Colorada Mine, Cananea Consolidated Copper Company. By WILLIAM CATRON and CYRIL U. COOLEDGE. (T.P. 938). 18
Some Outstanding Mine-hoisting Equipment. By BRUNO V. E. NORDBERG. (T.P. 1190, with discussion) 188
An Experimental Study of the Rock Drill. By T. SUZUKI. Digest by B. F. TILLSON. 206
Hot-milling of Rock-drill Bits. By ROBERT C. BERGGREN. (T.P. 1215). . . . 211
Recent Trend in Drill-steel Gauge at Homestake. By HARLAN A. WALKER. (T.P. 1214). 220
Air Transport at Agua Fria. By FELIX B. SHAY. (T.P. 1156, with discussion) 225
Hazards from Gases in Metal Mines and Protections against Them. By E. H. DENNY. (T.P. 984, with discussion) 230

PAGE

Safety and Health Efforts of the Anaconda Company at Butte. By JOHN L. BOARDMAN. (T.P. 993) 241

Ventilation and Air Conditioning of the Magma Mine. By C. B. FORAKER. (T.P. 979) 253

Ventilation and Dust Prevention in the Butte Mines. By A. S. RICHARDSON. (T.P. 969) 268

Air Cooling in the Gold Mines on the Rand. By WILLIS H. CARRIER. (T.P. 970, with discussion) 276

Suggestions for the Control of Silicosis in Mining. By DONALD E. CUMMINGS. (T.P. 930) 288

Index 309

COMMITTEES

Since the publication of the last Transactions volume on Metal Mining (Volume 126 in 1937), officers and members of the Mining Methods Committee and of the Committee on Health and Safety in Mines have been listed as named below:

Mining Methods Committee

Chairman

J. Murray Riddell, 1937–1939
Guy N. Bjorge, 1940

Vice-chairman

Guy N. Bjorge, 1937–1939
Clyde E. Weed, 1940

Secretary

Roland D. Parks, 1939
Harlan A. Walker, 1940

Members

Ralph S. Archibald
A. H. Bebee
Clinton Bernard
Guy N. Bjorge
Roy A. Bowen
C. P. Browning
Louis Buchman
P. B. Bucky
B. C. Burgess
Blair Burwell
J. J. Carrigan
Bert M. Concklin
Michael Curley
W. B. Daly
J. H. Davis
T. C. DeSollar
G. E. Diehl
S. H. Dolbear
G. J. Duff
R. B. Earling
Stanly A. Easton
Lucien Eaton
S. R. Elliott
Carroll R. Forbes
Arthur B. Foote
Darrell Gardner
F. H. Gartung
H. C. Goodrich
W. H. Goodrich
Benere H. Grant
E. A. Hamilton
P. G. Harrison
Burrell Hatcher
Earl E. Hunner
Charles F. Jackson
Verne D. Johnston
W. A. Knoll
John Knox
William Koerner
Walter S. Larsh
E. T. Lednum
R. D. Leisk
F. W. Maclennan
F. W. Nobs
H. R. Norsworthy
Arthur Notman
Ralph D. Parker
O. B. Perry
Roland D. Parks
C. A. Pierce
George W. Potter
J. B. Pullen
George S. Rice
J. Murray Riddell
C. M. Romanowitz
H. L. Roscoe
A. J. M. Ross
George H. Rupp
William H. Schacht
O. M. Schaus
Philip J. Shenon
Gerald Sherman
Frank J. Smith
J. B. Stapler
R. B. Tempest
B. F. Tillson
Charles Thurman
F. A. Wardlaw, Jr.
Leland A. Walker
Clyde E. Weed

Committees continued on next page

Committee on Health and Safety in Mines

Chairman

O. M. Schaus, 1937–1939 J. J. Carrigan, 1940

Vice-chairman

J. J. Forbes, 1937–1938 R. R. Sayers, 1939–1940

Secretary

L. E. Dick, 1937–1938 Carl M. Fellman, 1939–1940

Members

S. H. Ash
E. W. Bullard
J. J. Carrigan
M. D. Cooper
F. S. Crawford
E. H. Denny
L. E. Dick
Cadwallader Evans, Jr.
Thomas Fear
Carl M. Fellman
Milton H. Fies
J. J. Forbes
C. A. Garner
Oscar A. Glaeser
Daniel Harrington
Earl E. Hunner
A. J. Lanza
H. J. Mutz
Ralph D. Parker
G. B. Pryde
George H. Rupp
J. J. Rutledge
J. T. Ryan
R. R. Sayers
O. M. Schaus
Clyde E. Weed

Recent Improvements in Mining Practice on the Mesabi Range

By Arthur E. Anderson,* J. Murray Riddell,† Member A.I.M.E., and Grover J. Holt‡

(New York Meeting, February, 1938)

Out of the depths of each business cycle we emerge with a stimulus for greater efficiency and a realization of progress in industrial technique. The recent years have not been an exception to this rule, and in retrospect we sense that significant advances have been made in mining practices on the Mesabi Range, with emphasis on speed of production and economy of operation. A survey of such changes does not reveal anything that is fundamentally or basically new, but it is apparent that ideas and machines that have found wide application in industry elsewhere, with necessary modifications, are being put to productive use in mining iron ore. Innovations in transporting and conveying ore from the open pits stand out with greatest prominence, so that, in the main, it will be with these features that this paper will treat, in the review of progress made to this date.

The late years have witnessed the development of small deposits for open-pit operation, the cleaning up of pits nearing exhaustion, and in two large operations the approach to the limit for haulage by steam locomotives, owing to grades, making it necessary to recover the ore in slopes, track benches and below the power-shovel limit by a milling and scram operation.

For several years small standard-type dump trucks have been used in clean-up or scram operations within the pits, but during the season of 1936 the first effort was made to use heavy truck haulage in the transportation of stripping and iron ore. The new units in use today vary from 15 to 20 tons in capacity, and are equipped with powerful gasoline or Diesel engines and supplemented by trailers or wagons on rubber or crawler treads. The Diesel engine finds favor with most operators because of fuel economy. These units develop upward of 125 hp. in the 15-ton trucks. The speed of these trucks varies from 2 miles per hour in low gear to 35 in high gear. The transmission affords 12 speeds forward and 3 speeds in reverse.

Manuscript received at the office of the Institute Feb. 17, 1938. Issued as T.P. 968 in Mining Technology, November 1938.

* Oliver Iron Mining Co., Duluth, Minn.

† Assistant Manager, Republic Steel Corporation, Duluth, Minn.

‡ Butler Brothers, St. Paul, Minn.

As these units are mounted on pneumatic tires, and a cut or bruise resulting in a blowout entails expense, it is essential to have good roads. Roadways are easily built and maintained by means of tractors, bulldozers and blades. A useful and economical supplement to these mechanical means is a patrolman on foot equipped with a rake and pick.

TRUCKS AND TRACTORS

In truck service, the size of the shovel dipper used in loading should be restricted by the size of the tires and the shock that they are capable of withstanding. A 1½ or 2-yd. machine is better adapted to loading present trucks than a 4-yd. dipper, from the standpoint of shock as well as the distribution of load in the truck.

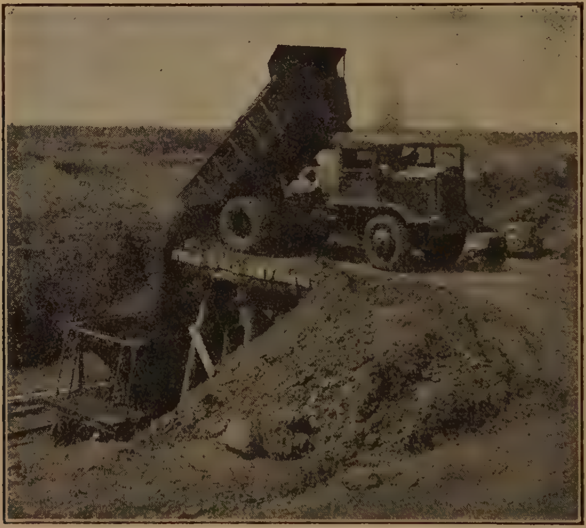


FIG. 1.—END-DUMP TRUCK WITH DUAL REAR-WHEEL DRIVE.

Various types of chassis are in use. One type is shown in Fig. 1, using dual rear-wheel drive mounted on 5¾-in. load-carrying axles, with the load distributed over six tires. The trucks shown in Figs. 2 and 3 are driven by four sets of dual rear wheels, providing traction on eight tires and distributing the load over ten tires. This distribution of load has not as yet proved its superiority over the straight dual type. These trucks also require a longer turning radius than the standard dual type. The latest type of truck unit to be placed in operation is the tractor truck with bottom-dump trailer (Fig. 4). The driving unit consists of a tractor truck with short wheel base, which affords a minimum turning radius. These trucks are of two types, the standard front-wheel steer and the four-wheel drive with four-wheel steer.

Various types of bodies are in use, suited to the particular problem of the operator. Fig. 1 shows an end-dump type. This 9-cu. yd. body is without an end door and is dumped by a single hydraulic hoist. It is especially adapted to dumping from ramps into cars, and for stripping service where the crest of the dump would be too soft for a roadway and therefore unsafe for use of side-dump trucks. It must, however, be backed up to the crest before dumping.

Side-dump bodies are in use on one-way and two-way dumps. In Fig. 2 a 9-cu. yd., one-way dump body is shown. This type has the disadvantage of being able to dump on one side only, therefore the approach to the dump or pocket must be from one direction only. The dumping is accomplished by two hydraulic hoists. Fig. 3 shows a 9-cu. yd. two-way side-dump body, which closely resembles a standard railroad dump car in its action. Any side-dump body allows a run around on the dump and is well suited to ore haulage with a properly designed pocket.

FIG. 2.—ONE-WAY SIDE-DUMP BODY WITHOUT HINGED DOOR.

The bottom-dump trailer (Fig. 4) is well adapted for dumping into pockets, but not so well to ramp service. In stripping, this deposits the earth in windrows, which are "dozed" off with a bulldozer. This unit has the advantage of rapid dumping and can be emptied at full speed when necessary, through doors that admit a clear opening 8 to 12½ ft. long and from 3 to 4½ ft. wide. These trucks have a capacity of 10 to 15 cu. yd. of stripping, and from 12 to 18 long tons of iron ore.

To date, tractor and wagon haulage has been limited to Diesel-powered tractors and wagons equipped with crawler treads. These wagons have varied from 8 to 16 cu. yd. in capacity, capable of dumping on either side by means of hydraulic hoists. In Fig. 5 is shown a typical stripping operation where the earth is being dumped in a mined-out area on the pit bottom. Here the short-turning radius is seen, the easy curve in evidence being one of convenience rather than necessity. This type of side-dump wagon has great stability due to the bearing on the track combined with the width between treads and can therefore dump more easily while traveling than the side-dump truck. The center of gravity is also somewhat lower. The condition of the roadway is not material to its operation, and it can load ore after a heavy rainfall. The tractors used have a top speed of 6 m.p.h., and in stripping on level hauls of 1500 ft. three of these units have moved from 1200 to 1400 cu. yd. per 8-hr. shift, with ease.

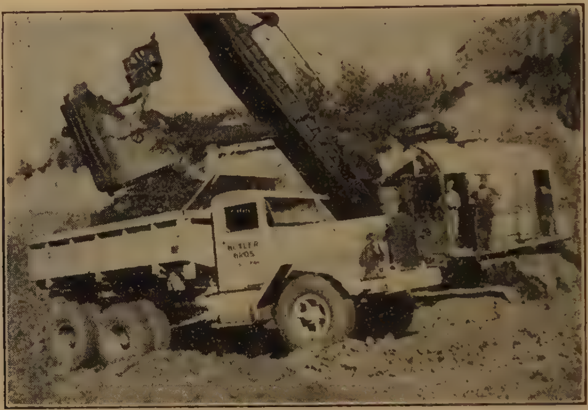


FIG. 3.—TWO-WAY SIDE-DUMP TRUCK DRIVEN BY FOUR SETS DUAL REAR WHEELS.

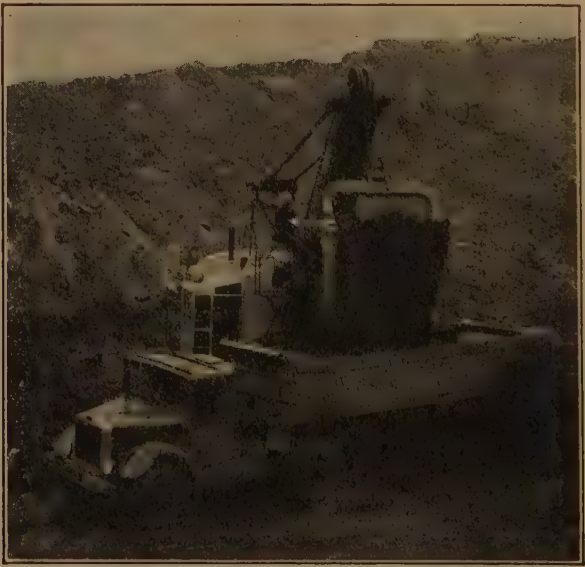


FIG. 4.—TRACTOR TRUCK WITH BOTTOM-DUMP TRAILER.

Experience so far with trucks and tractors has convinced operators that the tractor-wagon combination stands out in performance for short hauls of less than 1500 ft., while trucks excel for longer hauls, because their speed on return trips reduces the number of units required. A tractor requires less roadwork, although the repair cost on crawler pads is high on rocky roadways. Either unit has advantages over the conventional rail haulage, particularly for small pits; the ability to negotiate steep grades, 10 per cent and better; and curves of less than 50-ft. radius, eliminating switchbacks, makes available ore that would be tied up in track benches and at depths not attainable by locomotive haulage. The mobility of this equipment, without recourse to track work, is an important consideration.

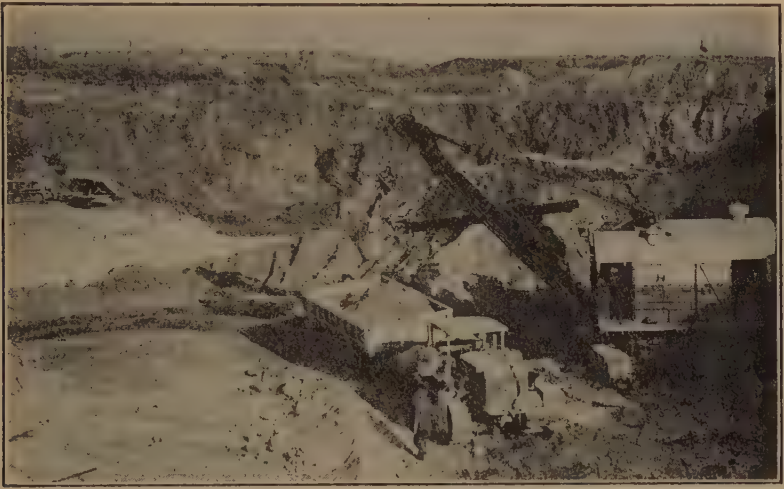


FIG. 5.—TYPICAL STRIPPING OPERATION EMPLOYING TRACTOR AND TRAILER EQUIPPED WITH CRAWLER TREADS.

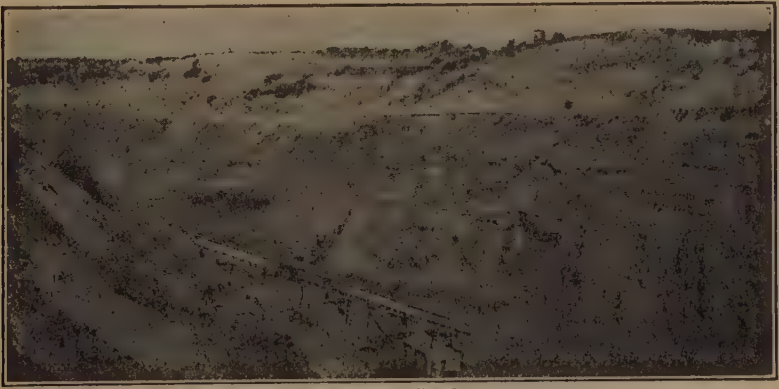


FIG. 6.—OPEN PIT, ST. PAUL MINE.

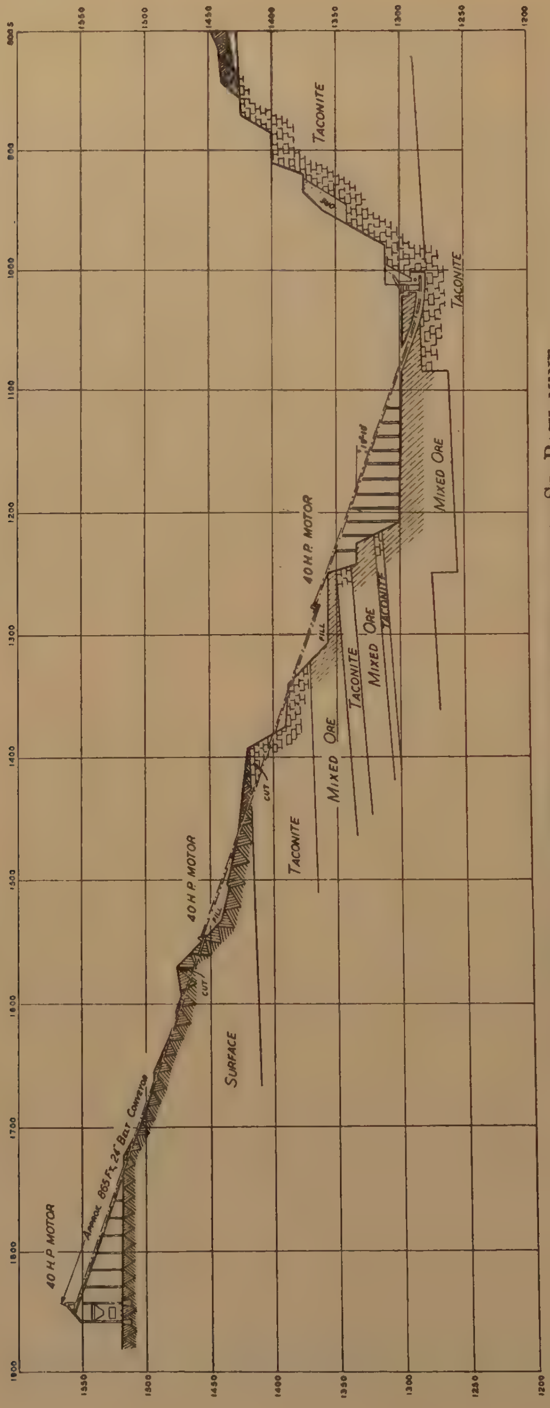


FIG. 7.—BELT ARRANGEMENT AND SCREENING PLANT, ST. PAUL MINE.

CONVEYOR-BELT SYSTEMS

Passing from transportation in which the vehicle moves with the load, we come to conveyance by stationary vehicles or movement by use of conveyor-belt systems. This means of transportation has been generally used in industry for a long time. The first large-scale application in mining was made by the H. C. Frick Coke Co. at the Colonial Coal Mines, in 1924. This conveyor is 4⅓ miles long. Later, in 1928, this same company installed a second system 3½ miles long to deliver coal to the Palmer dock on the Monongahela River. In 1935, conveyor belts for the movement of excavated material and aggregates for concrete were installed at the Grand Coulee dam, totaling in all well over 5 miles in length. This installation has been called "a construction classic."

The first application of conveyor-belt transportation in iron mining on the Mesabi Range was made in 1932 at the La Rue mine, near Nashwauk, Minn., by Butler Bros. This installation involves the movement of wash iron ore mined by underground methods to a washing plant on an adjoining property. It is necessary to stockpile in the open pit the ore mined during the winter months, when the washing plant cannot be operated. Belts are used to convey ore from the underground working faces to a hopper in the open pit. Transportation from this point to the stockpile is by electric motor haulage.

FIG. 8.—TRUCK DUMPING ON GRIZZLY, LOOKING ALONG LINE OF CONVEYOR BELT.

The lower portion of the belt, parallel to the stockpile, is 600 ft. long. The ore from the stockpile is loaded onto the belt by a 1½-cu. yd. gasoline power shovel through a hopper, which is movable along the length of the belt as the shovel progresses through the pile. The ore from the loading belt is transferred in turn to four sections inclined at 20°, to a bin near the washing plant.

During the summer months the current production is transferred directly to the loading belt.

This system employs 4440 ft. of 30-in. belt and has a capacity of 400 tons per hour. The horizontal belts are driven by 15-hp. motors at 440 volts; the inclined belts require 40 hp. at the same voltage.

The next application of conveyor belts to iron-ore mining in Minnesota was made by the Republic Steel Corporation at its St. Paul mine in 1936. This was formerly an open-pit operation employing steam-locomotive haulage, by which means a depth of 225 ft. had been attained on a uniform grade of 3 per cent. The problem confronting the operator was the removal of the remaining ore, largely locked up in slopes and track benches. The solution lay in three feasible plans: (1) to sink a shaft, drive a long drift in rock, install a hoist and erect a hoist building; (2) to install an incline along the slope of the pit bank, with a shaft at the bottom of sufficient depth to permit the loading of skips; (3) to employ a conveyor system. The second plan required the erection of a hoist and engine house. The third plan was adopted because the initial and operating costs were least (Fig. 6).

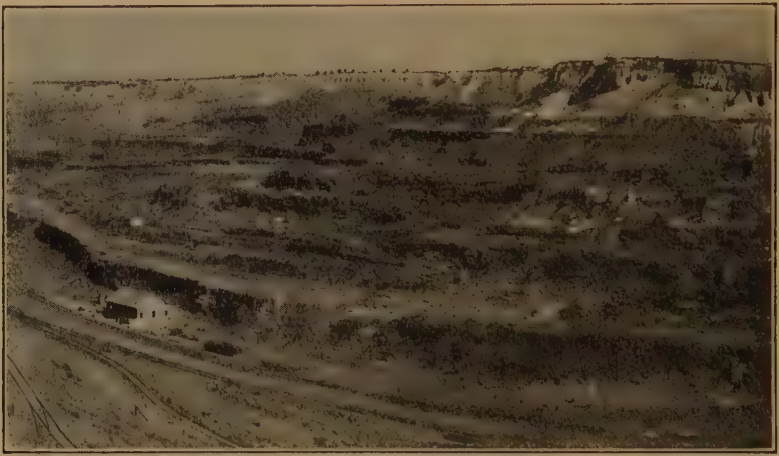
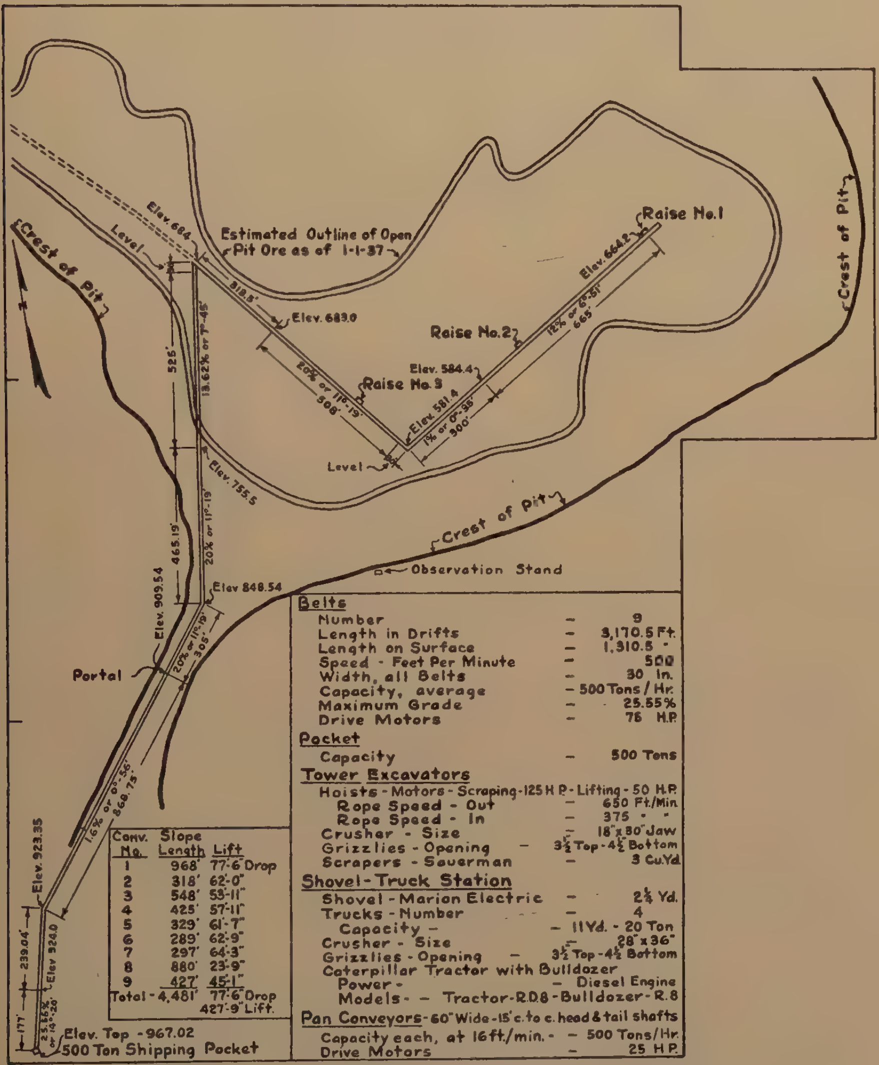


FIG. 9.—OPEN PIT, SPRUCE MINE, SHOWING MAXIMUM DEPTH ATTAINABLE BY LOCOMOTIVES AND ORE LOCKED UP IN TRACK BENCHES.

Two-way dump trailers of 13-cu. yd. capacity are loaded by 1¼-cu. yd. shovels driven by gasoline engines. The trailers are hauled by crawling tractor trucks to the lower end of the conveying system (Fig. 7). The trucks dump the ore directly onto a grizzly (Fig. 8), the fines falling into the loading pocket. The oversize passes to a crusher, from which the crushed product falls to the same loading pocket as the fines. The ore is fed onto the conveyor belt by a feeder with 9-ft. centers and 6-in. pitch. The feeder is capable of handling 250 tons of iron ore per hour at a speed of 25 ft. per minute. The conveyor belt transports the ore to a steel bin on surface, from which it is loaded into railroad cars if for direct shipment or into dump cars if it is to be washed.

The conveyor system has a length of slightly less than 900 ft. and is constructed in three sections of 284 ft. each, center to center, on an angle of 19°10′ from the horizontal. The belts have a speed of 350 ft. per minute and are driven by rubber-lagged pulleys 36 in. in diameter. The belts are 30 in. wide and of six-ply, 32-oz. construction. The head shafts are driven by 40-hp. gear motors at 2200 volts, the speed being reduced by chain and



Conv. No.	Slope Length	Lift
1	968'	77'-6" Drop
2	318'	62'-0"
3	548'	53'-11"
4	425'	57'-11"
5	329'	61'-7"
6	289'	62'-9"
7	297'	64'-3"
8	880'	23'-9"
9	427'	45'-1"
Total -	4,481'	77'-6" Drop 427'-9" Lift.

Belts
Number - 9
Length in Drifts - 3,170.5 Ft.
Length on Surface - 1,310.5 "
Speed - Feet Per Minute - 500
Width, all Belts - 30 In.
Capacity, average - 500 Tons / Hr.
Maximum Grade - 25.55%
Drive Motors - 75 H.P.

Pocket
Capacity - 500 Tons

Tower Excavators
Hoists - Motors - Scraping - 125 H.P. - Lifting - 50 H.P.
Rope Speed - Out - 650 Ft./Min.
Rope Speed - In - 375 " "
Crusher - Size - 18" x 30" Jaw
Grizzlies - Opening - 3½" Top - 4½" Bottom
Scrapers - Sauerman - 3 Cu. Yd.

Shovel-Truck Station
Shovel - Marion Electric - 2½ Yd.
Trucks - Number - 4
Capacity - - 11 Yd. - 20 Ton
Crusher - Size - 28" x 36"
Grizzlies - Opening - 3½" Top - 4½" Bottom
Caterpillar Tractor with Bulldozer
Power - - Diesel Engine
Models - - Tractor - R.D.8 - Bulldozer - R.8

Pan Conveyors - 60" Wide - 15' c. to c. head & tail shafts
Capacity each, at 16 ft./min. - - 500 Tons/Hr.
Drive Motors - 25 H.P.

FIG. 10.—BELT-CONVEYOR SYSTEM, SPRUCE OPEN PIT.

sprockets in the ratio of 12 to 37 teeth. The empty belt requires 4.2 hp. to start it, and 36.3 hp. when loaded. The carrier troughs are composed of three 5-in. removable pulleys, equipped with roller bearings.

INSTALLATION AT SPRUCE MINE*

The most recent installation of this kind was completed in 1937 at the Spruce mine (Fig. 9) of the Oliver Iron Mining Co., a mining subsidiary of the United States Steel Corporation. This mine is one of the older operations on the Mesabi Range, having produced to date more than 25,000,000 tons of iron ore. It has reached a depth in the open pit of 200 ft. This was about the economical limit for steam-locomotive haulage and the change to the milling and scram method of mining was imminent.

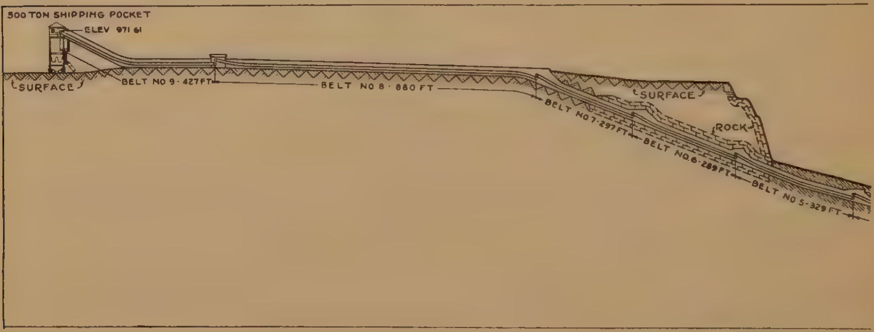


FIG. 11.—BELT-CONVEYOR

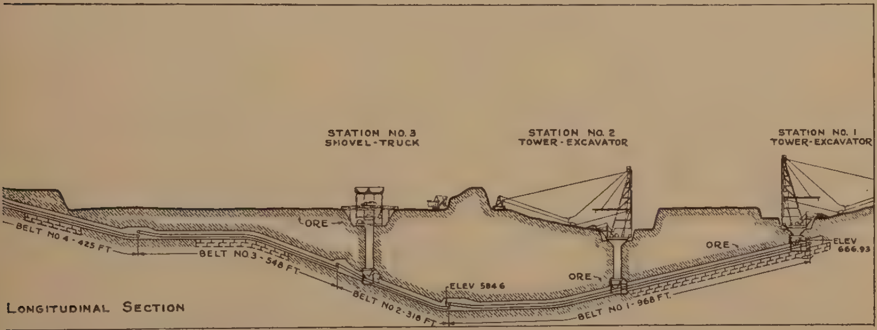
This method contemplated the use of a large number of raises extending from the lowest level of the mine to the bottom of the open pit, transportation by underground motor haulage, and hoisting through a shaft. The present layout at the Spruce mine is a radical departure from the original plan. The deviation is not so much in method as in the equipment used. It is still essentially a milling and scram operation.

In Figs. 10 and 11 are shown the plan and profile of the conveyor-belt system. Only one-third of the total length is above ground, the remainder being in a drift that closely follows the contour of the bottom rock, so that practically all of the remaining ore is tributary to the belt by gravity. There are but three raises, in contrast to a very large number contemplated in the original plan, and directly above raises 1 and 2 are two steel towers (Fig. 12), parts of a dragline excavator system. The ore passing through raise No. 3 (Fig. 13) is brought to the raise by means of trucks loaded by a 2¼-yd. electric revolving shovel. The trucks have a capacity of 11 cubic yards.

---

* An earlier description of this installation (by W. F. Schwedes and C. D. Skillings) appeared in *Skillings Mining Review* (Oct. 16, 1937).

The raises are 10 by 10 ft. in cross section, timbered and lined with 60-lb. rail. They are 100 to 120 ft. high and have a capacity of about 500 tons when full. As it is an essential requirement for trouble-free operation of a conveyor-belt system that the ore be devoid of large chunks, and to eliminate damage to cars, docks and boats, a small screening and crushing plant is installed at each raise. At No. 3 raise the trucks are driven over the raise and dumped (Fig. 14) on a bar grizzly 8 ft. long, set at an angle of 45°, with a top opening of 3½ in. The oversize is fed to the crusher by means of a pan-type feeder 60 in. wide and 15 ft. long. The feeder is driven by a 25-hp. motor at a speed of 16 ft. per min., with a resultant capacity of 500 tons per hour. The crusher is of the jaw type, 36 by 28 inches.



SYSTEM, SPRUCE OPEN PIT.

The towers above raises 1 and 2 are of steel frame construction, 100 ft. high (Fig. 15), and are set upon a circular track 30 ft. in diameter, so that they can be revolved 360°. The mechanism for operating the tower, scraper and crusher (Fig. 16), is in the base and the operator's cab is about midway up in the tower, affording a full view of the pit. The anchor towers (Fig. 17) are 18 ft. high, mounted on caterpillar treads, propelled by two 15-hp. motors, and are operated from the cab in the main tower. The anchor towers are ballasted by a weight of 40,000 lb. Both the traveler cable and the drag-scraper cables are operated by the Ward-Leonard system of control. The drag scrapers will operate within an average radius of 170 ft. but for clean-up purposes can be operated up to 700-ft. radius.

The drag scraper has a capacity of 3 cu. yd. and weighs 3200 lb. empty. It is crescent shaped and bottomless, equipped with cast manganese-steel teeth. As it fills, it becomes stern-heavy, which raises the front. When fully loaded, this action is sufficient to raise the teeth from their dragging angle, and the scraper rides on the backs of the teeth. It is dumped upon a scalping grizzly 8 ft. long with 16-in. openings. The material through this screen is transported by a pan conveyor, which empties upon a

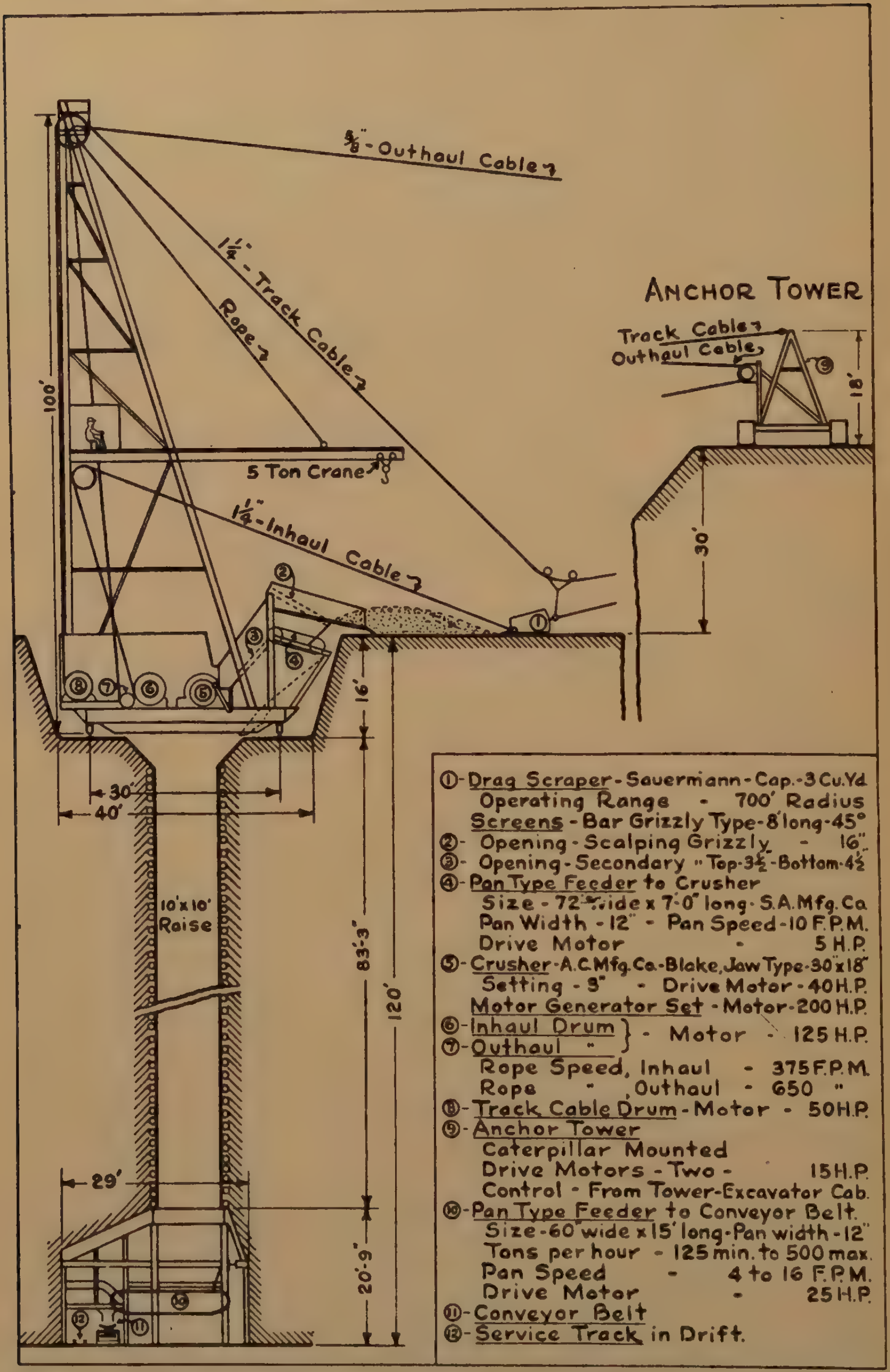

⅝"-Outhaul Cable
1½"-Track Cable
Rope
5 Ton Crane
1¼"-Inhaul Cable
100'
ANCHOR TOWER
Track Cable
Outhaul Cable
18'
30'
16'
30'
40'
10'x10' Raise
83'-3"
120'
29'
20'-9"
① ② ③ ④ ⑤ ⑥ ⑦ ⑧ ⑨ ⑩ ⑪ ⑫
①-Drag Scraper-Sauermann-Cap.-3 Cu.Yd
Operating Range - 700' Radius
Screens-Bar Grizzly Type-8' long-45°
②- Opening-Scalping Grizzly - 16"
③- Opening-Secondary "Top-3½"-Bottom-4½"
④-Pan Type Feeder to Crusher
Size-72" wide x 7'-0" long-S.A.Mfg.Co.
Pan Width-12" - Pan Speed-10 F.P.M.
Drive Motor - 5 H.P.
⑤-Crusher-A.C.Mfg.Co.-Blake, Jaw Type-30"x18"
Setting-3" - Drive Motor-40 H.P.
Motor Generator Set-Motor-200 H.P.
⑥-Inhaul Drum } - Motor - 125 H.P.
⑦-Outhaul "
Rope Speed, Inhaul - 375 F.P.M.
Rope " , Outhaul - 650 "
⑧-Track Cable Drum-Motor - 50 H.P.
⑨-Anchor Tower
Caterpillar Mounted
Drive Motors-Two - 15 H.P.
Control - From Tower-Excavator Cab.
⑩-Pan Type Feeder to Conveyor Belt.
Size-60" wide x 15' long-Pan width-12"
Tons per hour - 125 min. to 500 max.
Pan Speed - 4 to 16 F.P.M.
Drive Motor - 25 H.P.
⑪-Conveyor Belt
⑫-Service Track in Drift.

FIG. 12.—TOWER EXCAVATOR AND UNDERGROUND BELT FEEDER STATION, BELT-CONVEYOR SYSTEM, SPRUCE OPEN PIT.

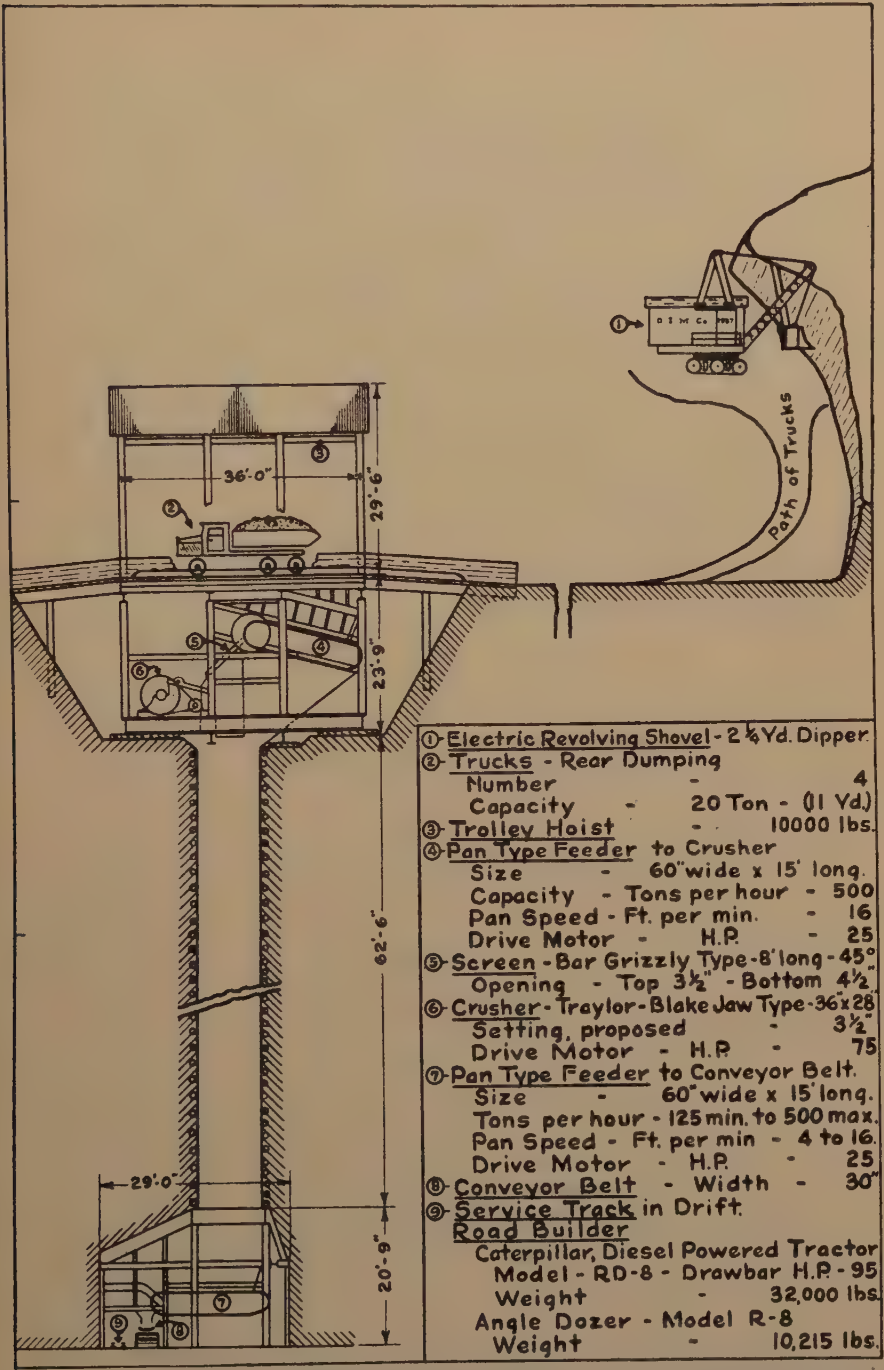

①
D I M Co.
Path of Trucks
③
36'-0"
29'-6"
②
⑤
④
⑥
23'-9"
62'-6"
29'-0"
20'-9"
⑦
⑨
⑧
①-Electric Revolving Shovel - 2¼ Yd. Dipper
②-Trucks - Rear Dumping
Number - 4
Capacity - 20 Ton - (11 Yd.)
③-Trolley Hoist - 10000 lbs.
④-Pan Type Feeder to Crusher
Size - 60" wide x 15' long.
Capacity - Tons per hour - 500
Pan Speed - Ft. per min. - 16
Drive Motor - H.P. - 25
⑤-Screen - Bar Grizzly Type - 8' long - 45°
Opening - Top 3½" - Bottom 4½"
⑥-Crusher - Traylor - Blake Jaw Type - 36"x28"
Setting, proposed - 3½"
Drive Motor - H.P. - 75
⑦-Pan Type Feeder to Conveyor Belt.
Size - 60" wide x 15' long.
Tons per hour - 125 min. to 500 max.
Pan Speed - Ft. per min - 4 to 16.
Drive Motor - H.P. - 25
⑧-Conveyor Belt - Width - 30"
⑨-Service Track in Drift.
Road Builder
Caterpillar, Diesel Powered Tractor
Model - RD-8 - Drawbar H.P. - 95
Weight - 32,000 lbs.
Angle Dozer - Model R-8
Weight - 10,215 lbs.

FIG. 13.—SHOVEL-TRUCK STATION AND UNDERGROUND BELT FEEDER STATION, BELT-CONVEYOR SYSTEM, SPRUCE OPEN PIT.

grizzly of the same specifications as at No. 3 raise, the oversize passing directly into the crusher, and the undersize falling into the raise. The pan feeder to the crusher is 72 in. wide and 7 ft. long. It is propelled by a 5-hp. motor at a speed of 10 ft. per minute.

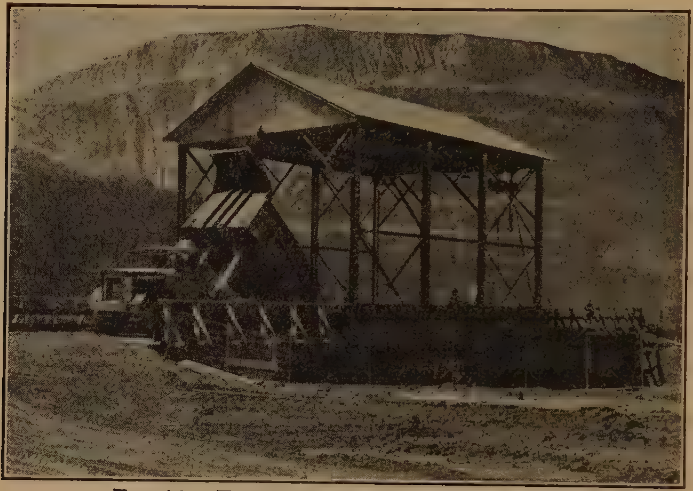


FIG. 14.—TRUCK DUMPING INTO NO. 3 RAISE.

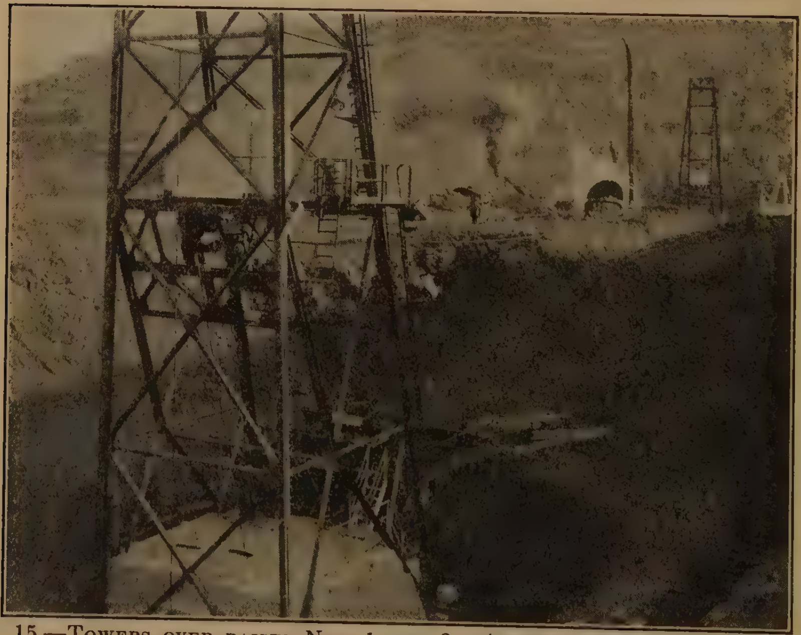


FIG. 15.—TOWERS OVER RAISES NOS. 1 AND 2. ANCHOR TOWER AND SCRAPER IN BACKGROUND.

The raises are hoppered on one side at the bottom, the weight of the ore in the raise resting directly upon the pan-type feeder to the conveyor belt. The feeders at each raise are similar, being 60 in. wide by 15 ft. long. They are driven by 25-hp. motors at speeds from 4 to 16 ft. per min. and deliver from 125 to 500 tons per hour to the conveyor belt, according to speed.

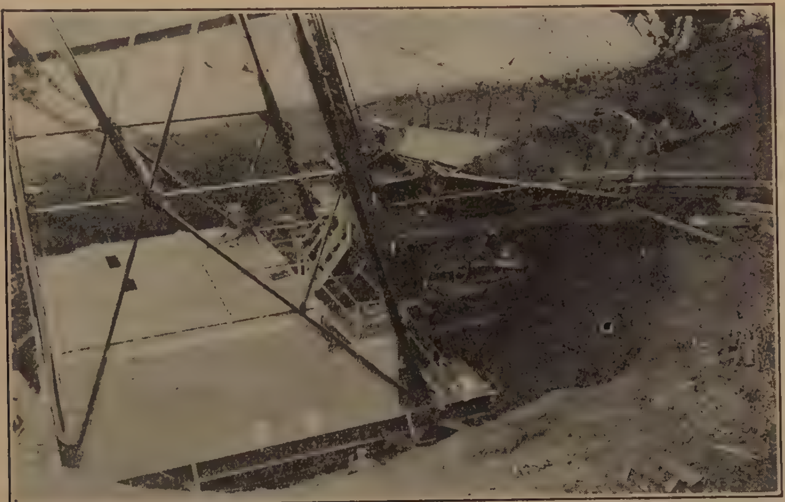


FIG. 16.—TOWER WITH SCRAPER ON IN-HAUL ABOUT TO DUMP ON SCALPING GRIZZLY.

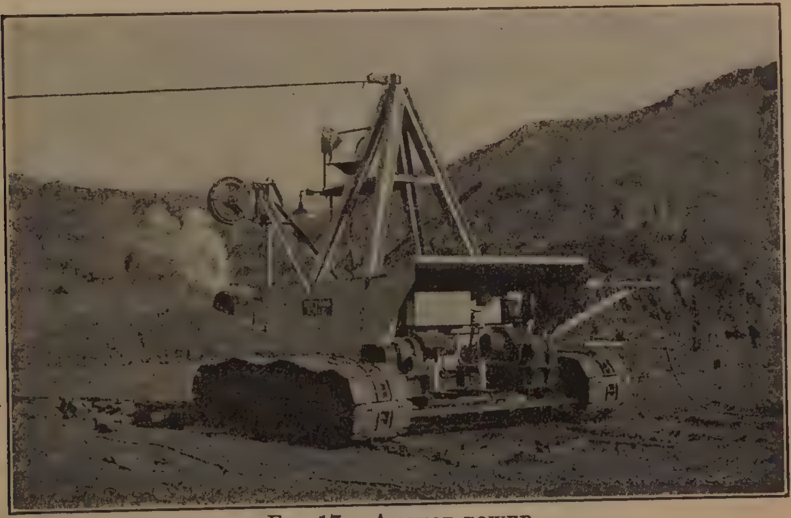


FIG. 17.—ANCHOR TOWER.

The belt-conveyor system is composed of nine sections, having a total length of 4481 ft. and a vertical lift of 386 ft. The belts travel at a speed of 500 ft. per min. and are capable of delivering on the average 500 tons per hour with a probable maximum of 750 tons per hour. Loading can be done at three points simultaneously by regulating the speed of the pan feeders.

The conveyor belt is 30 in. wide, seven-ply, 32-oz. duck, with rubber skin coats between plies. The top cover is $\frac{3}{16}$ in. thick, of highest grade rubber and $\frac{1}{16}$-in. bottom cover, for a total thickness of $\frac{11}{16}$ in. The belt joints are vulcanized.

The carrier rolls are spaced on 3-ft. 6-in. centers, except at feed and transfer points, where the spacing is closer and where the rolls are of soft rubber instead of steel. The lower or return rolls are spaced at 10-ft. 6-in. centers.

The belts are driven by 75-hp. motors, sequenced to start and stop at the proper moment. The lagged driving pulleys are 48 in. in diameter and are directly connected by a semiflexible coupling to totally enclosed speed reducers. Electric thruster brakes provide uniform deceleration and ratchet type holdbacks prevent loaded belts from coasting backward when stopped.

The belt from the portal to the loading pocket is housed in a galvanized steel gallery on a steel frame. Underground it travels in an entry timbered with ordinary mine timber, using 10-ft. caps and 8-ft. posts. The underground portion is paralleled by a 24-in. service track with rope haulage. The steel loading bin is in the railroad yard, and has a capacity of 500 tons of iron ore. The arc-type gates for loading, suspended directly above the railroad car, are operated by compressed air.

Accessory equipment, such as Diesel-powered tractors and angle dozers, are used for roadmaking and for other purposes.

Conclusion

This, then, briefly describes the salient features of the transition from laboring locomotives to a silent continuous flow of ore that is fascinating and intriguing. It is not possible to predict the extent to which these latest developments will be used. It is not probable that, in the larger operations, the use of standard railway equipment for haulage and power shovels for loading will soon be supplanted. Some major and many minor problems confront operators in bringing the use of trucks, tractors and conveyor belts to a balanced state of efficiency, but it can be confidently asserted that these will be overcome.

The successful completion of these innovations in iron mining reflects the skill and cooperation of many men, and, as well, the progressive management that maintains the reputation of Mesabi Range operators as masters in the art of moving earth.

DISCUSSION

(*Gerald Sherman presiding*)

T. T. Read,[*] New York, N. Y.—Is it economical, compared with carrying the tracks around, to take the conveyor a long way horizontally?

J. W. Van Evera,[†] Crosby, Minn.—There is only about 1300 ft. of the conveyor on the surface. This mine had been operated as a steam-shovel pit with locomotive haulage and the yard tracks were all in. Probably the pocket was near the established tracks. It would be cheaper to do that than to move the tracks to a pocket nearer the mine, because once a belt is loaded, the next 2000 ft. or so of level transportation does not add much to the cost. It would add a little to the construction cost.

T. T. Read.—Is it a general practice now to use dragline buckets without bottoms? I have not seen them before for surface work.

J. W. Van Evera.—This is an adaptation of the underground scraper. It is the only one that has ever been used on the surface on the Mesabi Range, to my knowledge.

T. T. Read.—It must save a good deal of wear.

J. W. Van Evera.—Yes, and it is easier to load with the kind of material to be handled than a bottom-type dragline bucket would be.

F. C. Ruhloff,[‡] Milwaukee, Wis.—It is also an adaptation of the excavation buckets that are used for levee building on the Mississippi River. The original drag buckets were built with bottoms. Then it was found that the scraper type (similar to the one described in this paper), merely a gathering bucket, was able to carry greater loads with less strain on the machines than buckets having bottoms. The friction of earth on earth is much less than the friction of the steel plates on the earth; old steam machines were increased in capacity by changing from the old type of digging bucket to a gathering type.

B. F. Tillson,[§] Upper Monclair, N. J.—Would not that depend upon the coarseness of the material?

F. C. Ruhloff.—Certainly, if the material were coarse, a cutting blade would be needed. But even where a cutting blade was necessary, it was found advantageous to leave out the bottom half of the bucket.

J. W. Van Evera.—I think the coarser material would require the teeth and the bottomless bucket. I do not think coarser material could be handled easily with a bucket having a bottom.

G. S. Rice,[||] Alexandria, Va.—Have you ever tried steel band conveyors?

J. M. Riddell.—As far as I know, a steel belt has not been tried.

---

[*] Professor of Mining Engineering, School of Mines, Columbia University.

[†] Manager, Gordon Mining Co.

[‡] Bucyrus Erie Co.

[§] Consulting Engineer.

[||] Consulting Mining Engineer.

McH. Mosier,* College Park, Md.—As I understand it, the Spruce mine is an old mine that was being reopened, so it was necessary to provide equipment. Under what conditions would it pay to scrap rail haulage in favor of belt haulage?

J. W. Van Evera.—The Spruce mine was in operation with locomotive haulage right up to the time that this new equipment was installed. In fact, in the last shipping season the Spruce mine operated almost entirely with locomotive haulage. In the coming shipping season it will operate entirely with the new equipment.

I do not think that anyone could state in a few words just when a point would be reached at which it would pay to scrap the older haulage and install the new equipment. In general, it comes at the time when the change of method is imminent; in other words, at Mr. Riddell's property, the St. Paul, a change in the method of mining would be necessary within a very few years. At this mine, I think they might have worked another three or four years with locomotive haulage but with constantly increasing costs because of continually higher grades.

B. F. Tillson.—Have you any correlative experience as to the maximum grade of belt conveyor and its speed, which you found satisfactory both from the point of view of keeping the load on the belt and not having too great a maintenance in wear of the belt?

J. M. Riddell (author's reply).—The Mesabi Range ores vary greatly in physical characteristics. There is a coarse, granular ore, and also very soft ores that are argillaceous or sticky, so the belt depends on the variations that must be met. It is natural that with the sticky ores the belt can be inclined at a greater angle than with other ores. In all the investigations that have been carried on in the work to date, in most tests an angle of about 19° appears to be the maximum—and there are special conditions where that would be too steep.

As far as belts are concerned, one can get into the question of belt economics, the speed range generally being between 350 and 500.

B. F. Tillson.—Do you think there is any correlation between the speed and the slope—that you can run at a higher speed with a flatter slope?

J. M. Riddell.—With a flatter slope you can run at a higher speed, other things being equal. The speed can be 500 ft. at a 19° slope, and that speed is probably close to the maximum with a soft, sticky ore.

Member.—Have any cleats been used with these belts for the higher ridges?

J. M. Riddell.—I think the cleating is more to combat the necessity of moisture condition in a rumbly ore than anything else. The tendency is for it to wash on account of the excessive wetness.

---

*U. S. Bureau of Mines.

A New Method of Top Slicing at Kipushi, Katanga, Belgian Congo*

By G. Van Esbroeck† and M. Van Weyenbergh‡

(New York Meeting, February, 1939)

The copper mines of the Katanga region in the Belgian Congo lie along the same mineralized belt as those of Northern Rhodesia. There are two distinct types of deposits in that belt, the dolomitic and the quartzitic. In the dolomitic type the metallic minerals are found in a dolomitic limestone country rock. Through oxidation and enrichment, large deposits of high-grade oxidized copper ores were formed, some of which are being worked by the Union Minière du Haut Katanga. Such deposits are all in Katanga.

The quartzitic deposits are found in Katanga and Northern Rhodesia. Here the primary mineralization occurred in sandstones and quartzites, in horizons stratigraphically lower than the dolomites. These rocks are not favorable to secondary enrichment and the oxidized zone is poorer than the sulphide zone, which alone contains the ore.

There are also in Katanga a few ore-bearing veins. One of them is being worked in the Prince Leopold mine at Kipushi.

Prince Leopold Mine

The Kipushi ore body fills a fault vein and invades the wall rocks more or less extensively by metasomatic replacement. The vein strikes north-northeast and south-southwest and dips to the west. The variable dip averages 60° to a depth of 300 ft., and more uniformly 80° below 400 ft. In the upper section the average width is 200 ft. and below the width is about 150 ft. The length of the ore body is 2300 feet.

The eastern side of the vein—the footwall—is dolomitic and contains irregular masses of ore sometimes extending 100 ft. or more from the main body. The hanging wall is a variety of calcareous schist, extensively mineralized within still more irregular limits.

---

* A French version of this paper appeared in the *Annales des Mines de Belgique* [vol. 39] but the matter being of especial interest to many members of the A.I.M.E. the Institute's Papers and Publications Committee waived its rule against duplicate publication in this instance. The courtesy of the Belgian Mine Administration in giving permission for this publication is appreciated. Manuscript received at the office of the Institute Aug. 20, 1938. Issued as T.P. 1078 in Mining Technology, July 1939.

† Professor of Mining, University of Ghent, Belgium.

‡ Manager, Mining Department, Union Minière du Haut Katanga, Jadotville, Belgian Congo.

There has been extensive movement on the vein marked by differences in the horizons of the footwall and hanging-wall rocks. Movement in the fault continued even later than the main period of mineralization, as shown by occasional brecciated areas, the elements of the breccia including at times very large fragments of country rock containing ore.

The rocks in both walls strike approximately east and west, thus being cut straight across by the lode. They dip from 40° to 70° toward the north. It is not impossible that the movement along the fault has been mainly horizontal and was connected with a well-known wide overthrust movement of the whole region west of the mine.

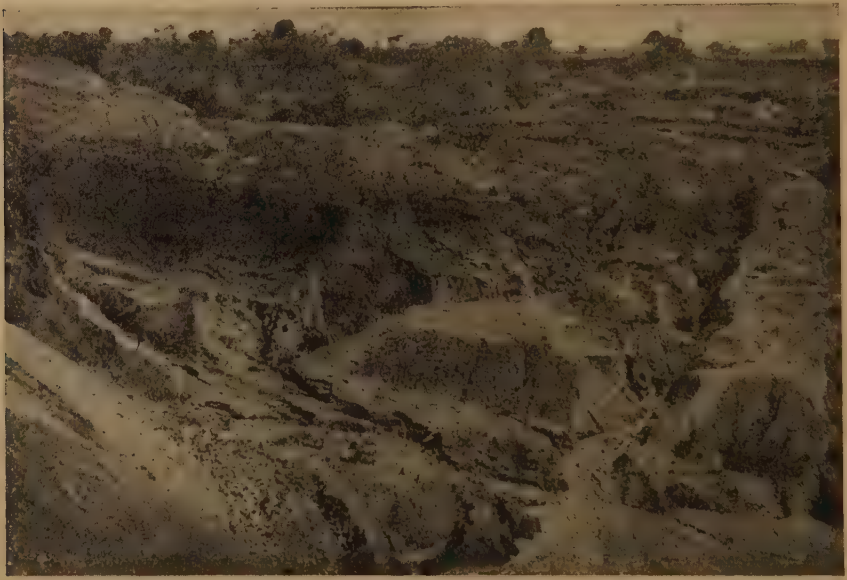


FIG. 1.—SUBSIDENCE AT BOTTOM OF OLD QUARRY.

The rich oxidized ores were mined down from the surface in an open-pit working to a depth of 100 ft., and were smelted in water-jacketed furnaces. The subsidence at the bottom of the old quarry is shown in Fig. 1.

At greater depth there is a gradual change to sulphide ores, containing bornite and chalcopyrite predominantly, with some chalcocite, the primary or secondary nature of which cannot be determined with certainty. Sphalerite is also found and some zinc ore is produced by selective mining. There is an appreciable quantity of silver in the sulphides, and native copper and silver are occasionally found.

It has been noted that oxidation may extend considerably below the present water table, possibly caused by deviously percolating meteoric waters, or a drier climate in earlier geologic times.

General Development

The locations of shafts and working levels are shown, with their relations to sections of the ore deposit, in Figs. 2 and 3. Two vertical working shafts 150 ft. apart were sunk in the dolomitic limestone, 250 ft. east of the ore croppings. They were sunk progressively, at first to a depth of 440 ft., later to 800 ft., and more recently to 1160 ft. for the main hoisting shaft and 1310 ft. for the service shaft. The advance of the service shaft will facilitate the development of future workings in depth, and the sinking of the hoisting shaft under a rock pentice when greater depth is needed.

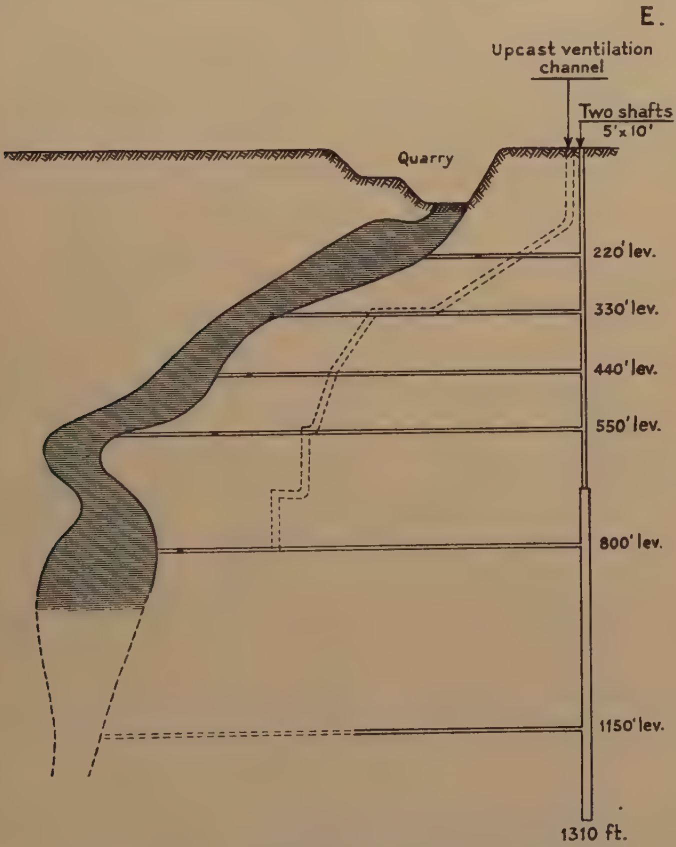


Fig. 2.—Cross section through Kipushi ore body.

Levels connect each shaft with the ore body at 110-ft. intervals between 220 and 550 ft. in depth. A fifth level at 800 ft. and the deepest at 1150 ft. have also been driven. Two shallow underground shafts connect the 670-ft. level with the 550 and the 800 levels. Similar connections will be made for a sublevel between the 800 and the 1150 level when mining begins below the 800, this sublevel to be connected with the hoisting and service shafts. At the 1150 level, the shortest horizontal distance from the shaft to the lode is about 750 feet.

The shafts have three compartments each, two being 5 ft. 9 in. by 4 ft. in useful area, and a third for ladders, pipes, etc., 5 ft. 9 in. by 3 ft. 3 in. in section. They are lined with reinforced concrete frames above the 550 level in the hoisting shaft and 700 ft. for the service shaft. Below these points the shaft sections are circular, without any lining, but subdivided into the same compartments. The cost of sinking is less, and that section is more efficient for the passage of ventilating air.

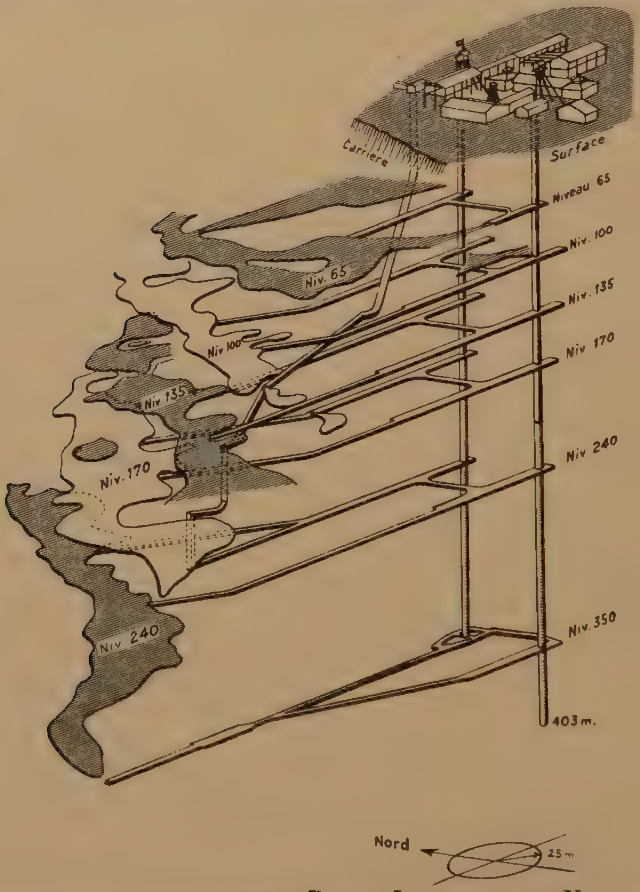


Fig. 3.—Underground development, Prince Leopold mine, Kipushi. Depths in meters.

A third shaft for ventilation is sunk in the footwall to the 800-ft. level on a varying inclination, to follow the lode at a horizontal distance of about 100 ft. It has a cross section of about 200 sq. ft. and is equipped with a ladderway to serve as an emergency exit.

At every level there is a crosscut from each shaft, connecting with the main footwall drift. Crosscut tunnels are turned at right angles to the main drift, at 50-ft. intervals, to locate the vein walls, and are usually driven some distances beyond the ore limits into the hanging wall.

Raises are driven from the crosscuts up into the ore and through it to the hanging wall, or mat, at distances apart of 35 to 40 ft., in some relation to the width of the ore.

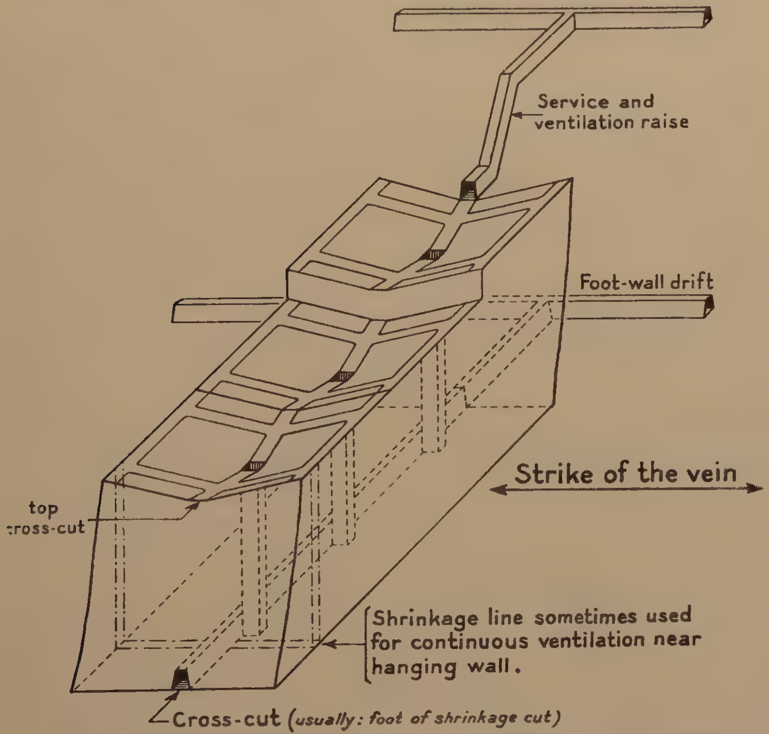

Service and ventilation raise
Foot-wall drift
Strike of the vein
top cross-cut
Shrinkage line sometimes used for continuous ventilation near hanging wall.
Cross-cut (usually: foot of shrinkage cut)

FIG. 4.—INCLINED TOP SLICING, WITH SQUARE SLICE ROOMS, EACH SECTION DIVIDED INTO BLOCKS.

Each crosscut is the axis of a stope section between two levels, bounded by two vertical planes 50 ft. apart parallel with the axis, and by the foot and hanging walls. A stope section is illustrated by Fig. 4, which shows the footwall drift, a crosscut and crosscut raises, and the layout of a slice floor in the ore with air connections for ventilation.

STOPING METHOD

The rather wide and irregularly lenticular ore deposit with a soft hanging wall and rich ore presented a difficult problem to the mining engineer. The ore is not only variable in outline, but must be separated into different classes according to its later metallurgical treatment, so that it can be sent to its proper destination:

1. Rich copper ore for direct smelting.
2. Ordinary dolomitic copper ores for the smelter. These ores give very poor results if subjected to concentration. The ore therefore must contain enough copper to pay its way through the smelter with a profit, or have some fluxing value.
3. Ordinary schistose ores for the concentrator.
4. Copper-zinc ores for differential flotation.
5. Copper-bearing zinc ores for special treatment.

The 500-ft. level, which is now the main producing level, yields about equal quantities of smelting and milling ores.

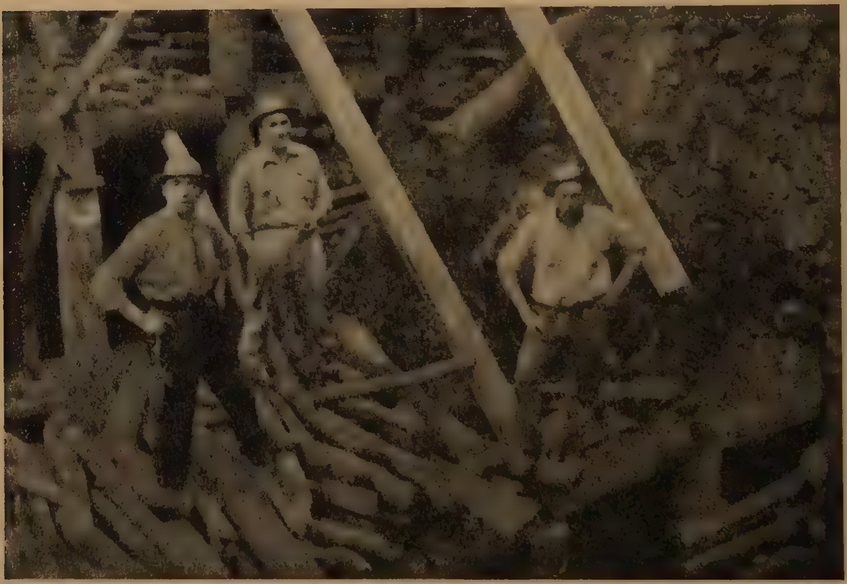


FIG. 5.—SQUARE-SET UNDERHAND STOPES, REACHING BOTTOM OF OPEN PIT IN 1929.

The first underground mining was done by square-set stoping (Fig. 5) beginning on the 220-ft. level and extending upward to the bottom of the open pit, which was reached in 1929.

In 1933 a section of the ore body was successfully attacked by top slicing. By 1934 it had been generally adopted in all stoping, which had then reached the 440-ft. level at some points. It was found that the crushed square-set timbers and waste filling made an ideal mat, protecting the slices, by subsiding slowly when the posts of the finished slice units were blasted down. When the ore was soft, a few posts could be recovered for a second use.

Restrictions brought about by the world depression made it necessary to select and mine certain blocks of ore, leaving other blocks behind in their original position. When slice operations in the blocks left standing were resumed, the faces against the blocks mined earlier exhibited furrowed slide planes typical of natural faults (Fig. 6).

Inclined top slicing was adopted in order to avoid horizontal tramming or scraping in the sublevel. The floor slope of 33° on which the ore would flow by gravity to the central crosscut raises was reduced to about 15°, the movement of ore being facilitated and at the same time being directed to the ore passes by iron-plate channels.

This reduced inclination has three advantages: (1) it was easier to sort the ore, as was often necessary; (2) accidents by falling rocks were less frequent, and (3) the large blocks could be broken on the spot, therefore a grizzly at the crosscut raise was no longer required.

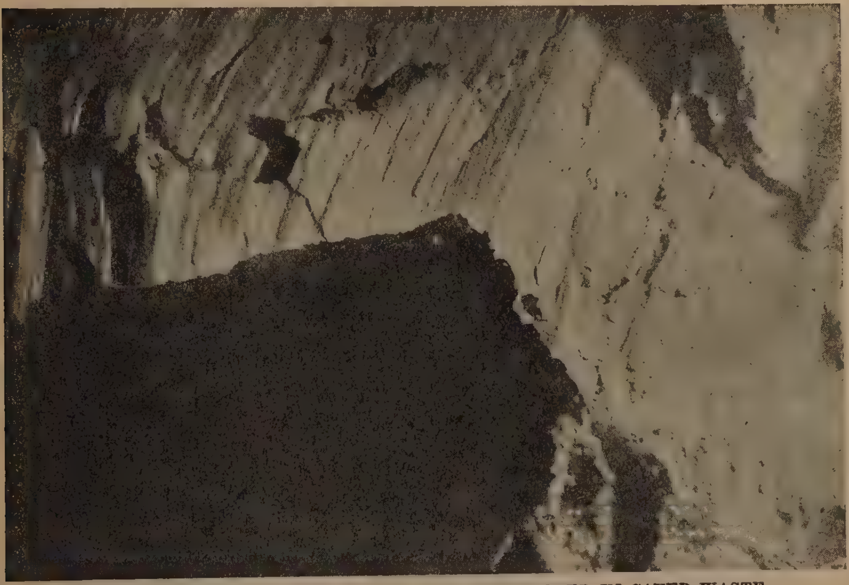


FIG. 6.—FURROWED SLIDING MIRRORS DEVELOPED IN CAVED WASTE.

The main feature in inclined top slicing is not the slope of the room, but the less extensive development workings in horizontal planes, and the more extensive development along vertical planes. This condition favors the use of gravity in moving the ore from the rooms down to the haulage level, thus reducing the total cost of mucking.

At first, narrow stopes timbered with square sets were mined upward over the development crosscuts on the vertical axial plane of each stope section, but it was soon found more convenient to raise them up as shrinkage stopes. Vertical timbered ore chutes were left in the shrinkage stopes every 40 ft. or so; two, three, four, or even six such chutes were used, according to the width of the ore. In very soft ore only the ore passes are raised through the ore, the space between them being left untouched. The shrinkage stope is driven to assist in starting the next slice below the mat and cannot be used when the ore is soft enough to collapse.

The crosscut in the slice floor, which parallels the development crosscut on the level, is not horizontal but slopes upward in each direction from the top of the raise to the unit room boundary, so that the ore can be more easily moved in the plate channels toward it. This scheme of inclined top slicing has developed into a new variation, each slope room sloping down from all sides toward the ore chute at its center.

When starting a new floor, the crosscut is not opened by drawing the broken ore down in the shrinkage cut. It is mucked by hand to the central chute. The shrinkage cuts, once established, are not used as ore passages, which thus eliminates the branch raises and sublevels as designed for the Humboldt mine at Morenci.

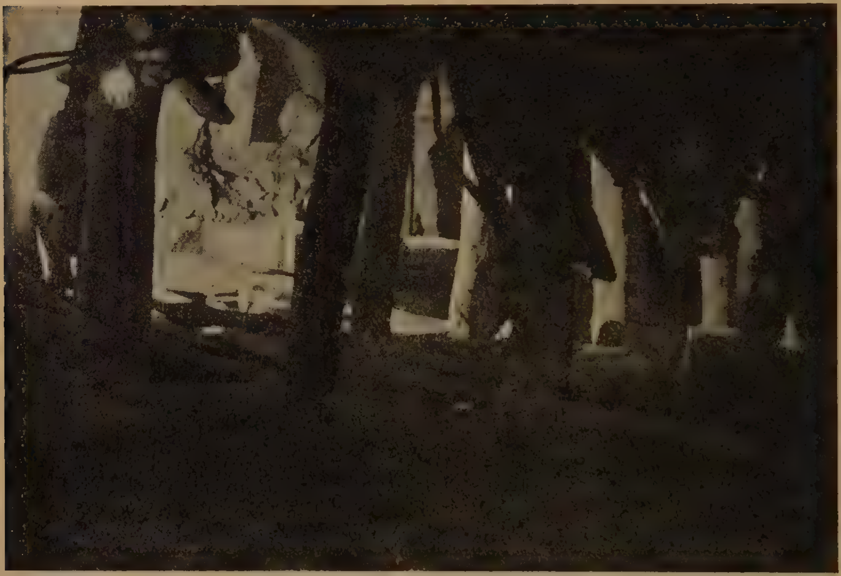


Fig. 7.—A room stoped out.
Horizontal or sill timbers serve as caps in next slice when mat is caved.

Another feature of special interest is provided by the retreating principle applied on a small scale in each slice room. At Morenci each room was stoped by starting at the central shrinkage stope and advancing upward toward its limit. At Kipushi each room is opened up by two sloping galleries about 6 ft. wide parallel with the strike of the ore, which start from the top crosscut at points about 12 ft. from the unit room boundary on either side. These galleries are united by crosscuts around the boundary of the slice unit block, making two side panels and one central panel on each side of the top crosscut, the central panel being left to protect the ore chute. The panels are thus mined in retreat.

This practice was adopted because of the high pressure exerted by the mat. It provides some relief when required by local conditions of softness of ore or subsidence. When the section is long enough to be divided into several rooms, which is usual, those adjacent to the hanging wall are mined first and work progresses toward the footwall. When production is to be increased, it may happen that one slice is attacked at its hanging-wall end before the slice above is completely finished at the footwall. When opening a room near the hanging wall, care should be taken to add new timber elements to complete the mat right against the hanging wall because, owing to its pitch, the ore extends beyond the timbers on the floor above.

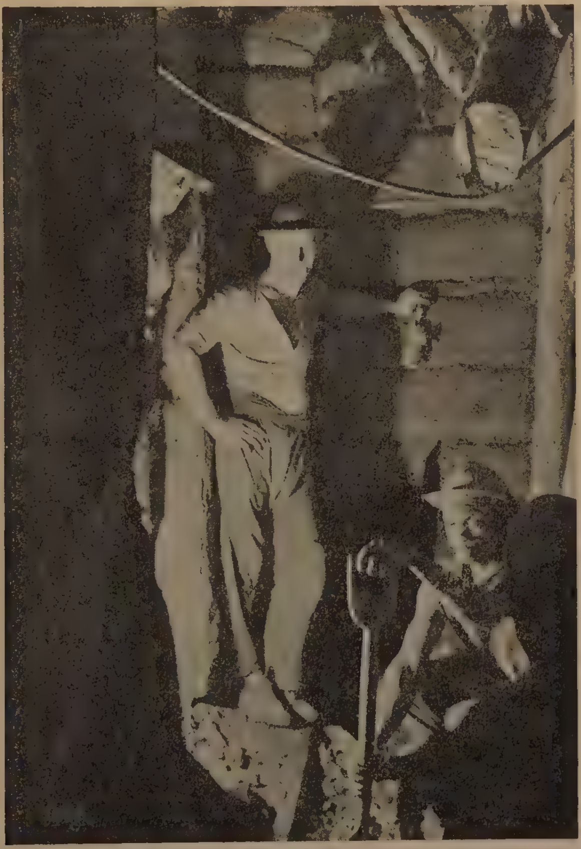


FIG. 8.—TOP OF ORE CHUTE WHEN A NEW ROOM IS BEING OPENED.
One side of cribs not yet pulled down. Mucking in an iron-plate channel.

Work is carried on in two contiguous sections at the same level as often as possible, so that the mat will not tend to break up. Slices are usually 7 to 8 ft. high.

When a room is stoped out, posts are drilled and blasted down, to promote the rapid collapse of mat and waste. Before this operation, new timbers are laid on the floor beside the posts, in small trenches cut in the ore for the purpose. When opening the next room beneath, these horizontal timbers are found undisturbed to serve as caps in the new slice (Fig. 7).

TRANSPORTATION

The ore in each room is shoveled or pushed toward the ore chute in iron-plate channels having a semicircular cross section. As the slope of an inclined stope is reduced from the angle of repose, 33° to 15°, it becomes possible to break up blocks of ore in the stope, and a grizzly at the ore chute is not needed.

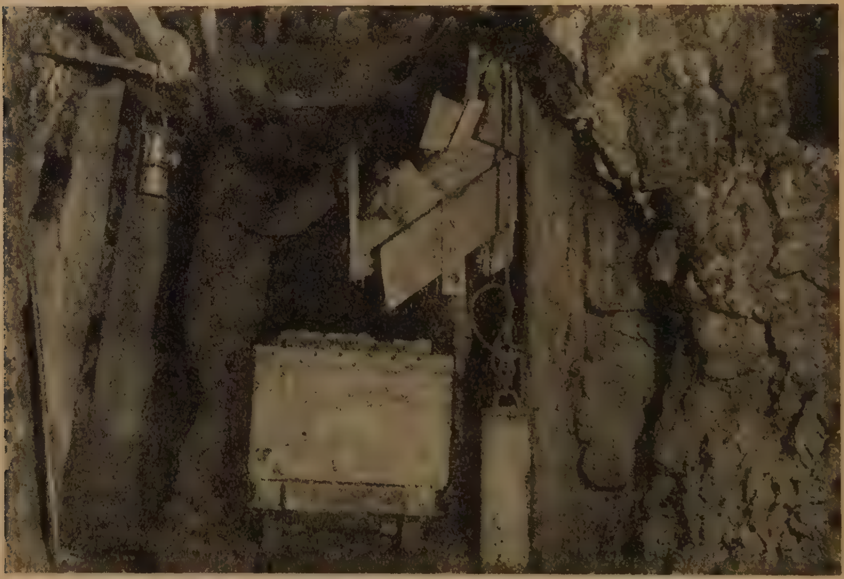


FIG. 9.—FOOT OF ORE CHUTE IN CROSSCUT OF STOPING SECTION.

The numerous development workings between the stoping and haulage levels of the Morenci inclined slices are not necessary at Kipushi, because of the flatter slopes of the floors in the rooms.

The upper openings of the chutes are protected by a few planks, which are easily removed and may be replaced in case of deformation.

The chutes are built of timber cribs lined with planks. They are supported on timber frames or bottom cribs set at one side of the crosscut (Fig. 8). The chute opening at the haulage level is controlled by a hinged door or a few planks (Fig. 9).

Hand tramming is used in the crosscuts and footwall drift. The cars are 2 ft. 8 in. high, about 4 ft. 3 in. long and 2 ft. 3 in. wide, with a wheel base of 1 ft. 6 in. They have capacities of 14 cu. ft. each, or of 1200 to 1800 lb. of ore. The wheels are fixed on the axles, which turn in grease-packed ball bearings.

The rails weigh 30 lb. per yard and are laid with a track gauge of 20 in. on treated sleepers of local hardwood, 3 ft. by 4½ in. by 4½ in., on 3-ft. spacing. The turnouts have a 33-ft. radius. At the intersection of a crosscut tunnel with the footwall drift, cast-steel turn sheets are used for switching cars off at right angles.

On the 670-ft. intermediate level, double inclined, gable-bottom cars are used. These are dumped by unlatching the side doors, which swing out on the upper horizontal hinges. The ore is dumped into an ore pass that leads to the 800-ft. level. Tramming on the main haulage level is done in trains of 20 to 30 cars from the ore passes along the footwall drift and the main crosscut. The trains are drawn by cables and compressed-air winding engines at a speed not exceeding 200 ft. per min. Since the haulage distance is short, more elaborate mechanization would not pay. On the 1150-ft. level the loaded cars will be moved by gravity. The main crosscut has been given a downward grade to the shaft suitable to that effect. The empty cars will be hauled back by cables.

HOISTING

There are two hoisting shafts, one for ore and one for service in handling men and supplies, and waste rock from prospecting and development work. Cages with four decks are used in the ore shaft and with two decks in the service shaft. A change from cages to skips for ore hoisting, to increase hoisting capacity, is being considered.

The two engines have 11½-ft. cylindrical drums with rope capacity of 1500 ft. and a rope speed of 1300 ft. per min. They are driven by alternating current, a 550-volt asynchronous electric motor, of 600 and 400 hp., respectively, for ore and service shafts.

The brakes are mechanical and are operated by hand.

The ropes are locked coil wire ropes 1⅓ in. in diameter, having a breaking strength of 75 tons each. The 11½-ft. diameter head sheaves are situated 85 ft. above the shaft collar and in pairs 4 ft. 8 in. apart.

Safety equipment consists of stops in the headframes and on the indicator dials, which automatically cut off the current and set the brakes to prevent overwinding or movement in any direction beyond a safe stopping point.

To prevent the cage from falling in the shaft after a rope-break from overwinding, "safety clutches" are set in the headframes 60 ft. above the columns.

VENTILATION

The air for ventilation through the mine is induced by two exhaust fans, an Aerex horizontal helicoidal fan acting against a vacuum of 7 to

8 lb. per sq. ft. and a centrifugal Pratt ventilator situated at the collar of the ventilation shaft, kept as a stand-by. The inlet air is drawn down through the working shafts.

The equivalent mine opening is 50 to 60 sq. ft. The normal quantity of air circulated through the mine is about 200,000 cu. ft. per minute.

As the stopes usually have no natural outlet for air circulation, and the slow combustion of the timber mat gives off heat, the ventilation of a top-slicing operation is unusually difficult. At Kipushi a strong air current is forced through the footwall drift to the farthest crosscuts, and then passes up through the manways which usually accompany the ore chutes to the floors. (Fig. 10.) It then goes through the top crosscut nearest the footwall, and through a ventilation and service raise to the next level above, and to the ventilation shaft.

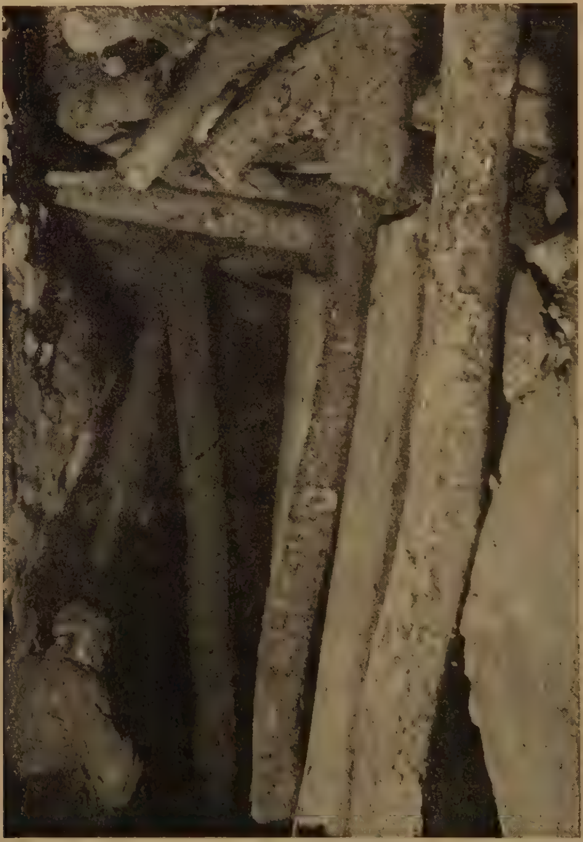


FIG. 10.—AIR PASS CONNECTING TWO OPENING GALLERIES OF CONTIGUOUS STOPING SECTIONS.

The ventilation is poor in the slice rooms during development, but when the sloping galleries are connected with the crosscuts the ventilation becomes satisfactory except in the rooms near the hanging wall. The slice floors are kept as much as possible on the same level in adjacent sections to facilitate air circulation.

If the ore outline on the hanging wall is not too irregular, a shrinkage stope is worked up longitudinally close to the hanging wall, from an elevation of about 15 ft. above the level to the working slices, to connect with the top crosscut and side galleries of the rooms. These shrinkage stopes may connect with several or many stope sections near the hanging wall and help to form a more extensive ventilating system on that side. These methods of working up from below are necessary because of the soft hanging wall in which it is practically impossible to maintain ventilation connections.

PUMPING

The ground-water level is found at a depth of 280 ft. The limestones and dolomites have a typically cavernous structure, with occasional free channels for water circulation that are assumed to connect with the vein fault. Such underground chambers may discharge great volumes of water until they are drained. The total flow of water from the mine increases as mining progresses and new areas are opened up by exploration. For the past two years it has been fairly constant at about 7,000,000 U.S. gallons per day at a depth of 800 ft. It went as high as 11,400,000 gal. per day for a few days when pumping was first started at that level. Under these conditions extreme precautions must be taken to avoid flooding the mine.

The pumping station on the 800-ft. level is equipped with five centrifugal pumps each capable of discharging 350 cu. ft. per min., or 3,800,000 gal. per day. They are driven by three-phase, 6000-volt electric motors. They force the water to the surface through one, or both, of two 16-in. diameter pump columns. The power current is sent down through two cables, one in each of the vertical shafts.

The pumping stations and shafts are isolated from the mine by concrete dams with steel doors (Fig. 11), which can be quickly closed in each main crosscut on every level below the water table. Two pipes closed by valves are built into and through each dam, which are used to drain the chambers behind them as desired. Since the rocks of the footwall of the vein are hard, the leakage around the dam is unimportant. To prevent water stored behind the dam for a considerable period from seeping through and creating pressure in a joint or seam between the concrete and rock, a few 1-in. holes flaring outward are drilled from the outside of the dam through the concrete into the rock.

A dam with a water door is constructed as soon as the crosscut is driven a few yards from the shaft station, and the chamber behind is subjected to a hydraulic pressure test before driving is continued to the ore body.

Whenever a development face is driven on a level below the pumping station, strict rules are imposed to avoid flooding:

1. An advancing face is accompanied by holes drilled with diamond bits in the same direction as the crosscut, to discover the nature of the rock and drain any abnormal water reservoirs.

2. In addition to the usual blasting holes, four to six test holes are drilled beyond any advancing face to depths of 20 ft., which slightly diverge into the future walls of the heading, and at least 8 ft. beyond the holes to be blasted.

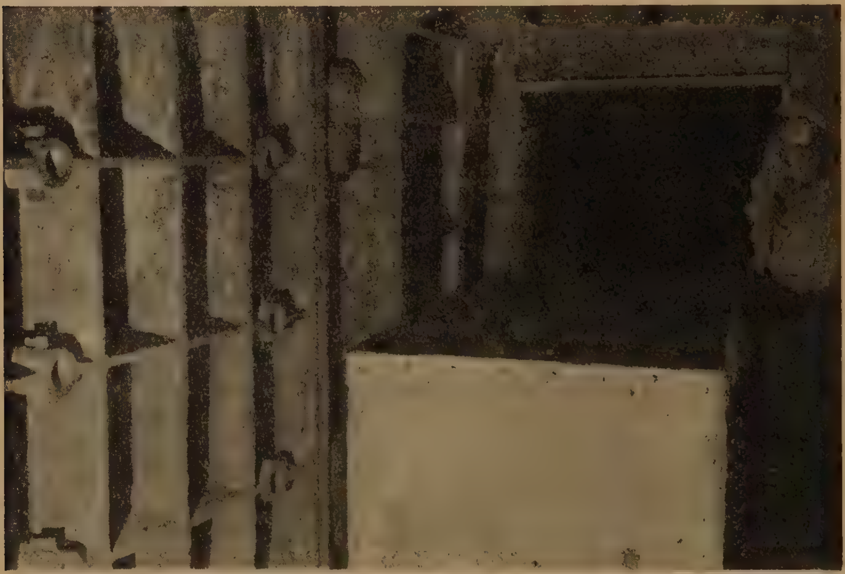


Fig. 11.—Water-tight steel safety doors in main crosscut at 550 feet.

3. Temporary water doors are built behind any advancing face, set in concrete frames or dams, which are removed and rebuilt more or less frequently according to the importance of the work to be protected. These doors are partly closed when blasting so that a strong rush of water would close them.

All dams and doors are designed and constructed with great care, as the safety of the mine depends upon them.

Power and Light

Alternating current at 55,000 volts three phases and 50 cycles is furnished by a branch line of the main 120,000-volt power line from the hydroelectric plant of Mwadinghusa, owned by a sister company of the mining enterprise.

The normal consumption of power is 2,500,000 kw-hr. per month, distributed as follows: hoisting and tramming engines, 180,000 kw-hr.;

air compressors, 400,000; ventilation fans, 70,000; pumping stations, 1,000,000; crusher and concentrator, 850,000; total, 2,500,000.

A converter station on each level produces 100-volt direct current, which is carried for lighting, etc., to all mining operations underground, including the stopes. The boys carry no lamps. The foremen and white men carry acetylene lamps with which to lead the boys in case of an emergency.

ORGANIZATION

The underground operation is directed by the general manager, an underground manager, two mining engineers (one for stoping and one for development work), two chief bosses (one for each shift), and seven bosses. The hoisting department and remaining underground personnel number 53 white men and 1550 boys. The workshops and ore-dressing department occupy 45 whites and 850 boys.

The work in the stopes is divided among units or gangs. A unit consists of one European miner and 30 to 40 boys, according to the size of the stope section. One unit may handle half of a stope section, but generally considerably more, and might cover two sections.

If the miners of the day and afternoon shifts comply with all regulations, they share a bonus computed on the total production of the unit, which is checked at daylight, each car bearing a distinctive mark.

The daily wage of the boys increases with the number of years of service at the mine. They are assigned to tasks by the miner.

Development work is run by white men, each directing 15 to 20 boys. They get a bonus for footing.

The mine is closed from 7 o'clock on Sunday morning until 5.30 on Monday morning. A watchman goes down on Sunday morning. The fan stops at 1 o'clock on Sunday until Monday morning. The pumps run through continuously under the attention of one white man. The pumping station receives enough ventilation from the moving parts of the motors themselves.

The monthly production of the mine varies greatly. For the time being, an average would be between 55,000 and 65,000 tons of ore.

DISCUSSION

(Gerald Sherman presiding)

H. A. LEIDICH,* Pottsville, Pa.—The inclined slices at the Kipushi mine consist of rectangular unit blocks about a central raise from which the ore is pushed in semicircular metal channels on slopes of 15° toward the raise. This method was developed: (1) to facilitate the sorting of ore in the stopes; (2) to reduce accidents resulting from falls of ore; (3) to enable large boulders of ore to be broken on the spot instead of on the grizzlies at the top of the raise.

---

* Safety Inspector, Philadelphia and Reading Coal and Iron Co.

The Kipushi method differs from the slices at Morenci in the slope of the slice floor; 15° in four directions from raises spaced at 35 to 40-ft. centers and timbered through a narrow shrinkage stope. Also, a panel is cut completely around a block 40 by 50 ft. before the stope is opened. In the Morenci method (Fig. 12), the slope

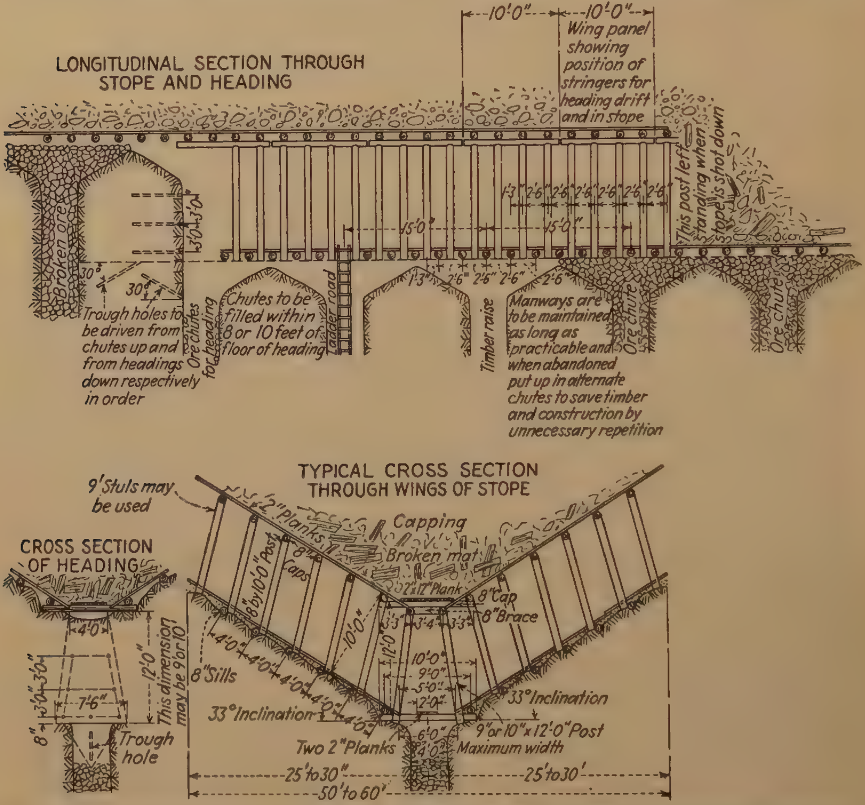

LONGITUDINAL SECTION THROUGH STOPE AND HEADING
10'0"
10'0"
Wing panel showing position of stringers for heading drift and in stope
This post left standing when stope is shot down
Broken ore
3'0" 3'0"
15'0"
15'0"
1'3" 2'6" 2'6" 2'6" 2'6" 2'6" 2'6"
30°
30°
Trough holes to be driven from chutes up and from headings down respectively in order
Ore chutes for heading
Chutes to be filled within 8 or 10 feet of floor of heading
Ladder road
1'3" 2'6" 2'6" 2'6" 2'6"
Timber raise
Manways are to be maintained as long as practicable and when abandoned put up in alternate chutes to save timber and construction by unnecessary repetition
Ore chute
Ore chute
TYPICAL CROSS SECTION THROUGH WINGS OF STOPE
9' Stuls may be used
2" Planks
Capping
Broken mat
8" Caps
8" by 10'0" Post
2"x12" Plank
8" Cap
8" Brace
10'0"
12'0"
3'3" 3'4" 3'3"
10'0"
9'0"
5'0"
2'0"
4'0" 4'0" 4'0" 4'0" 4'0"
8" Sills
33° Inclination
33° Inclination
9" or 10" x 12'0" Post
6'0"
4'0"
Maximum width
Two 2" Planks
25' to 30'
25' to 30'
50' to 60'
CROSS SECTION OF HEADING
4'0"
7'6"
3'0" 3'0" 3'0"
8"
12'0"
This dimension may be 9' or 10'
Trough hole

Fig. 12.—Typical incline slicing method for Longfellow mining division, Morenci, Arizona.

Headings to be advanced to extend over four chutes before starting of panel incline wings and to be run as small as consistent with timbering. No unnecessary ground must be broken.

Timbering of headings must always be kept as close to the face as possible.

Posts must be placed directly below cross sills of floor above. Sills to be bedded on solid ground, the ground being prepared by the pick or if necessary with the aid of small pop shots.

Longitudinal sills to be laid in each succeeding slice beneath the sill above so that joints break in the same vertical plane.

Panel wings to be carried up to 10 ft. wide upon an inclination of 33°. Posts in panels to be maintained in line and spaced to receive mat boards without necessity of trimming or fitting between.

In starting panels they must go up square from the heading and pains taken to preserve clean sharp corners.

of the slice floor is 33° in two directions, from raw raises, spaced at 15-ft. intervals and slotted at the bottom of the slice floor in order to establish gravity flow of ore to the raises.

Stoping progresses from one end of the slice to the other, parallel with the line of raises. The mined area is abandoned and can be shot down whenever necessary

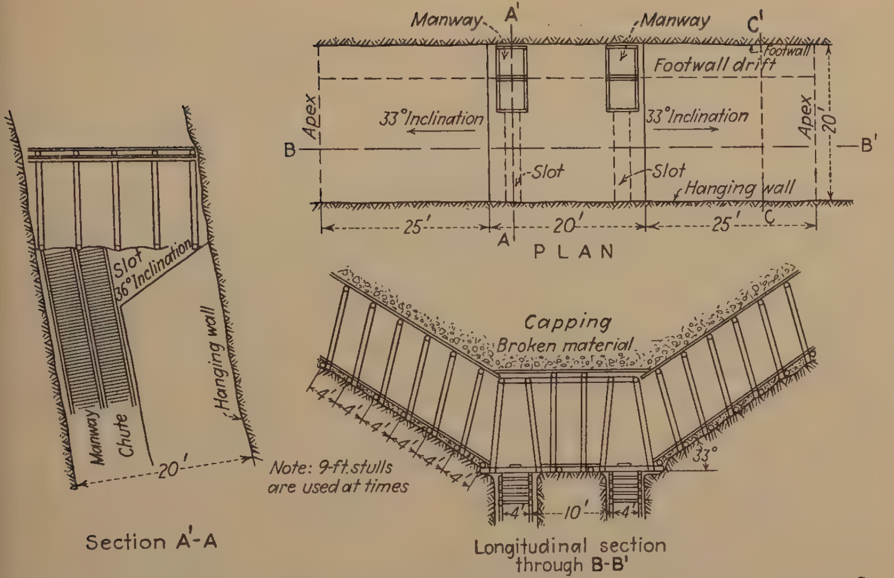


FIG. 13.—TYPICAL INCLINED TOP-SLICING METHOD OF MINING THIN VEINS IN OLD DOMINION MINES.

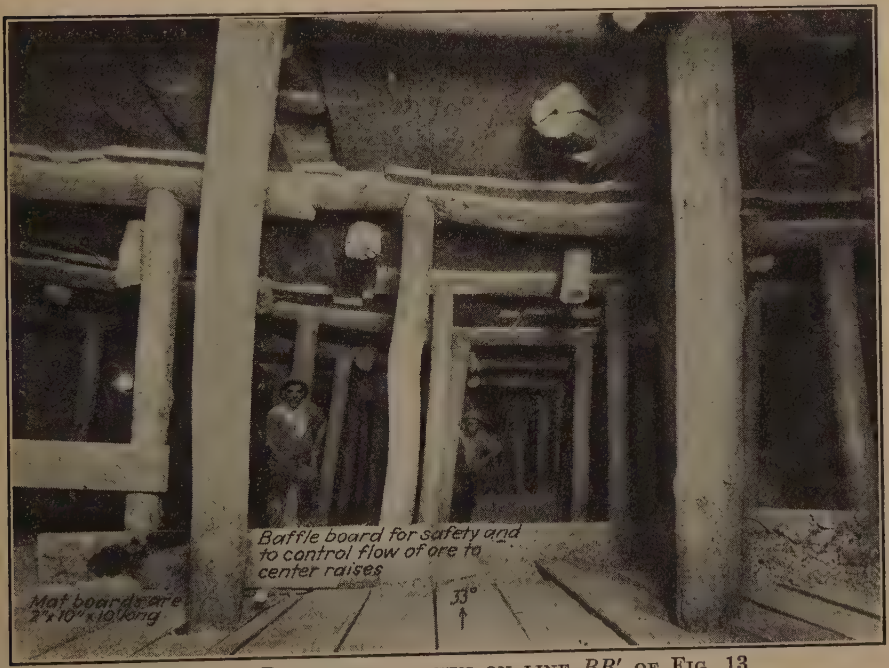

Baffle board for safety and to control flow of ore to center raises
Mat boards are 2"x10"x10' long
33°

FIG. 14.—PHOTOGRAPH TAKEN ON LINE *BB'* OF FIG. 13. Photograph by Gerald Sherman.

or advisable and another slice started beneath. The general practice is to allow the last row of panel props to stand until the new and adjoining panel has been completed.

The only sorting would be to separate waste and ore. Unless the waste is left in the stopes it is inconvenient to handle in transportation and hoisting. If stored in the stope it is apt to cause a break through the mat into the next floor below. As all material contains copper, it is cheapest to let it go to the mill with the ore.

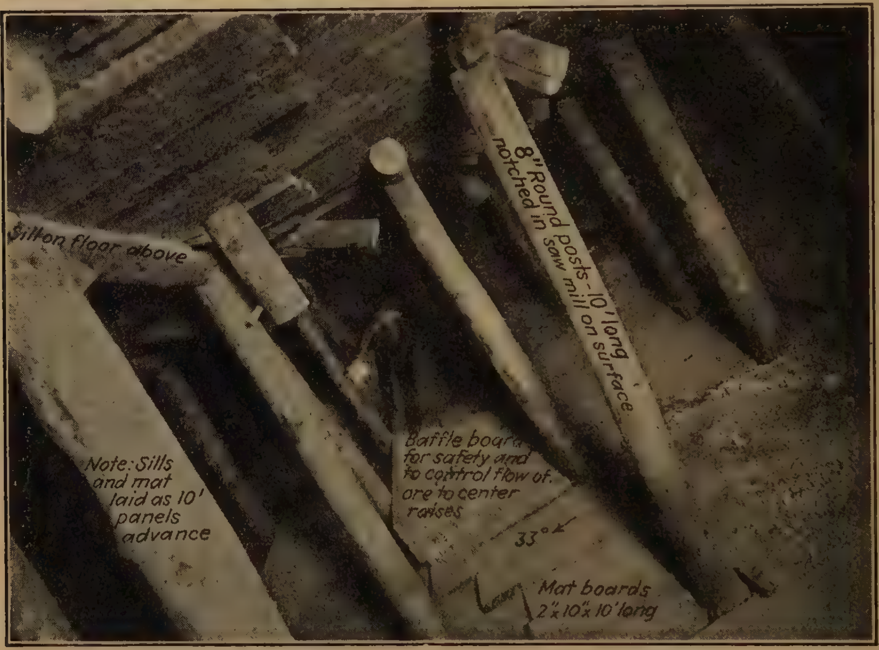

Sill on floor above
8" Round posts - 10' long notched in saw mill on surface
Note: Sills and mat laid as 10' panels advance
Baffle board for safety and to control flow of ore to center raises
33°
Mat boards 2"x10"x10' long

Fig. 15.—Photograph taken on line *CC'* of Fig. 13.
Photograph by Gerald Sherman.

The bottom of the slice, or heading, across the top of the raises is level, which facilitates the handling of supplies, also provides a safe approach and retreat to and from the active stoping area. Mat boards (Figs. 14 and 15) are placed across the panel to check the flow of ore to the chute; also to form a floor for miners to stand on while advancing the panel.

Table 1.—*Old Dominion Company—Globe, Arizona*

Method	Percentage Mined in Years Indicated					Tons per Man-shift	
	1926	1927	1928	1929	1931	1928	1931
Square setting	70	60	40	20	2	5.7	5.7
Shrinkage	15	15	15	14	10	15.0	15.0
Open stopes	10	5	5	3	2	6.4	6.4
Flat top slicing	5	10	2	2	1	7.5	7.5
Inclined top slicing	0	3	4	5	10	9.5	9.5
Morenci slide caving	0	7	34	56	75	17.3	26.0

TABLE 1.—(*Continued*)

OTHER DATA

	1926	1927	1928	1929	1931
Grade of copper in ore, per cent	3.9	3.8	3.6	3.3	2.0
Direct mining cost per ton	$5.56	$5.22	$4.82	$4.67	$2.24
Direct mining cost per pound copper	0.0803	0.0864	0.0955	0.1025	0.0706
Timber cost per ton	0.35	0.22	0.12	0.08	0.04
Lost-time accidents, average per month	7.3	4.9	1.3	0.8	1.7

In the ore at Morenci and the Globe-Miami districts, Arizona, it would cause excessive pressure if a panel were cut completely around a block 40 by 50 ft., preparatory to opening the slice. In retreating to a central raise in four directions, as at Kipushi, a dangerously heavy weight would be thrown on the gradually contracting pillar at the raise.

A modified method of inclined slicing (Figs. 13, 14, 15) was applied to the vein at the Old Dominion mine at Globe in 1927, yielding 9½ tons per man working in the stopes for an 8-hr. shift. This method was gradually replaced by more modern caving methods, with corresponding improvement in stoping efficiency and cost per ton, as shown in Table 1. The method used at Kipushi is satisfactory evidently for conditions there, because of the possibility of sorting the ore and because labor is cheap and cost of pushing the ore on the 15° slope is low.

L. EATON,* Milton, Mass.—The new method of inclined top slicing at Kipushi possesses some novel features, and marks an advance in the development of this kind of mining. In former practice the principal disadvantages of inclined top slicing have been that (1) the slope of the slices, 30° to 35°, made movement about the stope difficult and precarious, and increased the cost of erecting timber enough to counterbalance to a large degree the saving in the cost of mucking; (2) except in strong ore the use of the central slot as a chute was not satisfactory, because the sides squeezed in on the broken ore and blocked its flow.

By reducing the slope to 15°, much of the difficulty caused by the sloping floor has been removed, and some of the advantages of the slope have been retained, because the Kipushi ore apparently slides readily in the steel chutes. It must still all be shoveled, however. This is reflected in the tons per man-shift, which apparently are only 1.6 for the "boys" and 45 for the whites. This is higher than on the Rand but not nearly as high as in Rhodesia.

The difficulty of maintaining the long slot has also been overcome by leaving it full of ore and cribbing up chutes through the broken ore, a long slot not being necessary if the ore is conveyed in steel chutes.

Inclined top slicing was tried out in the soft hematite ore of the Marquette Range in Michigan some years ago, but was discontinued because of the disadvantages mentioned above. It might have been continued had it not been that scrapers came into use for sublevel work at about that time. By using scrapers the broken ore was put into the chute as cheaply as with the inclined slices, and the disadvantages of timbering and working on an inclined floor were dispensed with.

The African native quickly becomes an expert scraper operator, and excellent results should be obtained by using scrapers at Kipushi. Another great advantage would be a reduction in development work. Crosscuts could be driven at 200-ft. intervals instead of 50 ft., and the distance between raises need not be less than 100 ft., each raise thus serving a block of ore 200 by 100 ft. It might, however, be advisable to put up raises with three compartments, in order to provide two chutes. This would allow the use of two scrapers at each chute, and would facilitate separation of grades of ore. To mine a block of ore as large as this the scraper hoists should be driven by 20 or 25-hp. motors, and hoe-type scrapers, 42 or 48 in. wide with small side plates should be used. The ore would be mined by radial slices, starting at one boundary and swinging around the raise. By taking a pillar 100 ft. wide on each side of the raise, the longest pull would be 110 feet.

---

* Consulting Engineer.

If Lake Superior practice is any criterion, a production of 40 to 50 tons per shift could be expected from each scraper outfit.

The change in mining practice would also be an aid to ventilation. The Lake mine, at Ishpeming, Mich., had about the same capacity as Kipushi, and the ore was mined by top slicing. By using a fan with a capacity of 45,000 cu. ft. per min. at a pressure of 15 lb. per sq. ft. (3¼ in. of water) and controlling the flow of air by a judicious use of mechanically operated doors this mine was thoroughly ventilated, best results being obtained when the sublevels were arranged in steps. The main fan maintained a flow of air between chutes on the sublevels, and distribution to the face, when necessary, was accomplished by small inspirators and small secondary fans, but these were seldom needed.

From the figures given in the description of Kipushi practice, 200,000 cu. ft. per min. of air through a 50 to 60-ft. opening, the velocity of the air current would be nearly 4000 ft. per min., and this could not be obtained with a fan that provides a pressure of only 8 lb. per sq. ft. without a very high percentage of short-circuiting. Closer attention to leaks in the air circuit and a higher-pressure fan would probably help the ventilation in the stopes materially.

Rope haulage is very popular in Africa, and remarkably low costs are obtained, especially at the diamond mines at Kimberley, but even with cheap native labor and ore that breaks fine hand tramming and rope haulage with ¾-ton cars cannot compete with large cars hauled by locomotives. The disadvantage of the small cars is not only in the tramming. It reacts on every operation all the way to the face. Small cars require small chute openings, which will not pass large pieces, and so sledging is increased and production is reduced.

Conservatively it seems that 4-ton cars, preferably of Granby design, or rocker dump, and storage-battery locomotives are indicated for Kipushi. Of course, this would require the substitution of skips for cages, but that should be done anyway. If the skips and cars are of the same capacity, and if a single pocket of like capacity is built underground for each skip, the different grades of ore can be kept separate as easily as with cages. There are so many different grades of ore to be handled that a storage pocket is out of the question. Apparently the present hoist will handle 4-ton skips in balance, and hoisting capacity will be ample to a depth of 1500 ft. or more. Few changes would be necessary underground, and these could be easily made; but the changes necessary in the headframe might be more serious.

Preliminary Stripping of the Morenci Open Pit, Arizona

By Walter C. Lawson*

(Tucson Meeting, November, 1938)

The first plans were made in 1930 for the mining by open-pit methods of the low-grade disseminated ore body now known as the Morenci open pit. It was not until 1937, however, that final plans were completed and the decision reached to go ahead with the actual development.

Many advances were made in open-pit mining equipment during that period of seven years, therefore the ways and means of attack actually used were very different from those originally contemplated. Diesel-operated bulldozers, big trucks, full-revolving electric shovels and electric churn drills constitute the major equipment finally purchased for this modern stripping operation. Neither the bulldozers nor the big trucks were considered in the early operating plans, for at that time bulldozers had not been used in mining operations and the trucks as yet were undesigned. The full-revolving shovel and the churn drills were not new developments in open-pit equipment, but both had undergone remarkable manufacturing transition, emerging far more efficient and far more dependable than their predecessors. Even the ordinary drilling steel had been fundamentally changed. It was now threaded so that the dulled cutting bits could be detached and replaced, thus eliminating the costly transportation of long bars to the drill shop for resharpening.

The decision to use trucks in the preliminary stripping operations was based on the desirability of this type of transportation to economically prepare the pit for rail haulage, coupled with the success that the contractors had had with them on other big excavation jobs. There are at present no plans for the use of trucks beyond the preliminary work, inasmuch as rail haulage will give lower costs after roadbeds and benches are once established.

For many years to come, both ore and waste haulage will be over favorable grades. The average length of ore haul between the pit and an assembly yard will be about two miles. From there the crushers will be 1⅓ miles with a connecting track grade of 0.4 per cent. The upper ore benches of the pit will be reached from the yard by a series of four switchbacks over a ruling grade of 4.0 per cent compensated for curvature. Tracks to the dumps will be on 0.2 per cent grades.

---

† Manuscript received at the office of the Institute July 18, 1938. Issued as T.P. 980 in Mining Technology, September 1938.

* Phelps Dodge Corporation, Morenci, Arizona.

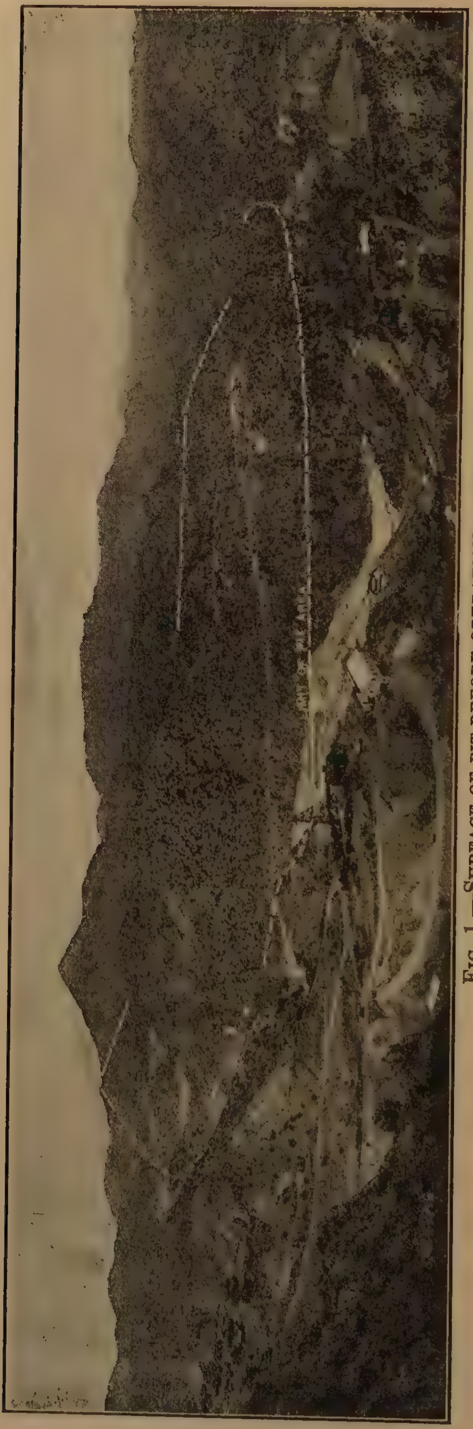


FIG. 1.—SURFACE OF PIT BEFORE OPERATIONS BEGAN.

The topography surrounding Morenci is extremely rough and mountainous (Fig. 1). The physical position of the ore body and the proposed site of the crushers is such as to bring a series of precipitous bluffs and deep canyons on the alignment of the track between the two. In like manner, tracks to the dumps will cross rugged country. The use of trucks, it was reasoned, would permit the operation to be started quickly and would allow the dumping of waste according to predetermined plans, so as to fill across the canyons encountered in track construction. It was evident that this procedure would greatly facilitate the construction and reduce the cost of putting rail into the mine. That is to say, instead of deferring waste removal until rail was worked into the mine, the plan was to start the stripping and work out of the mine.

Since the type of mining contemplated was new in the district and the camp had been on a close-down basis for a number of years, a great deal of preliminary engineering and several weeks of preparatory construction work necessarily preceded the actual movement of the first dipper of rock by the stripping shovels.

Laying Plans

The Morenci deposit that is now being made available by open-pit mining contains copper minerals, principally chalcocite, disseminated in porphyry. The ore body has been explored by diamond drilling and underground openings and is estimated to contain 284,000,000 tons above milling grade assaying 1.036 per cent copper with relatively unimportant amounts of gold and silver.

The mining limits are based on a pit layout (Fig. 2) contemplated to extract about 230,000,000 tons of ore carrying 1.06 per cent copper. This represents a reduction in tonnage through increase in grade cutoff limits and exclusion of portions carrying high stripping ratios. Final pit lines are projected on 45° over-all slopes.

Pit studies of tonnages, copper content and overburden showed that the mining of ore is justified through a vertical range of 850 ft., and that the relative elevations of the ore body and the crusher location permit the movement on favorable grades of 85 per cent of all material to be handled. Moreover, in the recovery of this ore it is estimated that 239,000,000 tons of barren or slightly mineralized waste will be removed, making a ratio of 1.04 waste to 1.00 of ore. The average thickness of the capping overlying the ore is 216 ft., with a maximum thickness of 500 feet.

Obviously, this sort of an ore body requires a large amount of advanced stripping in order to expose sufficient ore for steady mill production. Obvious as well is the necessity for keeping the advanced stripping to minimum requirements, so as to bring the mine to the productive stage without undue capital expenditure.

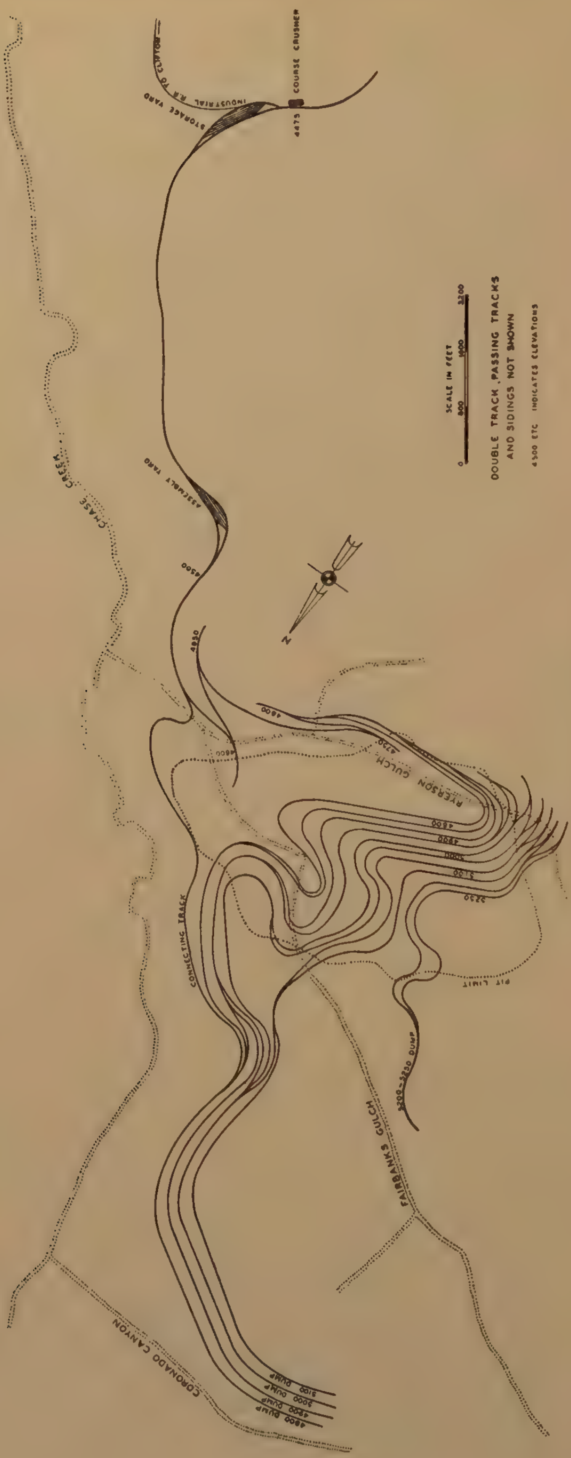


FIG. 2.—LAYOUT OF PIT.

Bearing in mind the foregoing as well as the desirability of making an initial attack that would develop a mine having good physical properties and would readily conform to expansion beyond the preliminary stage, an area nearest the proposed mill location was selected as being the most economical for the preliminary stripping. Further consideration in the selection of this area was that its development permitted the main approach tracks to the mill and the service tracks to the upper waste benches to be put in their permanent position from the outset. (Fig. 3.)

The establishment of banks 50 ft. high was early decided to be most economical. Since benches, after once being opened, must be left in position to be mined back as necessary, provision was made in the plans and layouts for the maintenance of bench widths of 100 ft. Based on all of these considerations and specifications, the limits of the preliminary stripping stage were defined on the maps and marked out on the ground, and the initial production has been and will be confined within these boundaries.

One of the foremost considerations in the investigation of the feasibility of mining an ore body by open-pit methods is the availability of sufficient dump room. Engineers' figures show that a canyon that is tremendously big to the eye will soon be filled in by a big stripping operation. Also, it must be remembered that, because of the swell from shooting, a 50 per cent greater volume is needed for rock moved from place into a dump.

Morenci is reasonably well off for dumping space. About one-half of the total stripping will move in a direction distinctly opposite from the ore movement and over tracks built especially for the handling of waste. The remainder will travel to dumps over the tracks used for the delivery of ore. The longest waste haul will be about three miles and the average will be slightly less. Under consideration is the possibility that a portion of the mineralized capping may be hauled to especially prepared dumps for leaching.

Adjoining the pit boundaries on the north is a sizable gulch that is physically well suited for the disposal of much of the preliminary stripping. The haul to this gulch is short and the approaches favorable, therefore it was particularly desirable for dump ground. Even though the geology of the area is unfavorable for the existence of ore, it was deemed necessary to prospect this ground before covering it up with waste, because of its proximity to the pit. A diamond-drilling contract was let and the subsequent completion of 10 holes averaging 800 ft. in depth disproved the area of ore.

The drilling was overlying and adjoining a porphyry-granite contact. The ground proved to be so badly shattered and broken up that core recovery was extremely low, but in anticipation of the likelihood of finding these conditions, which would make dependable sampling difficult

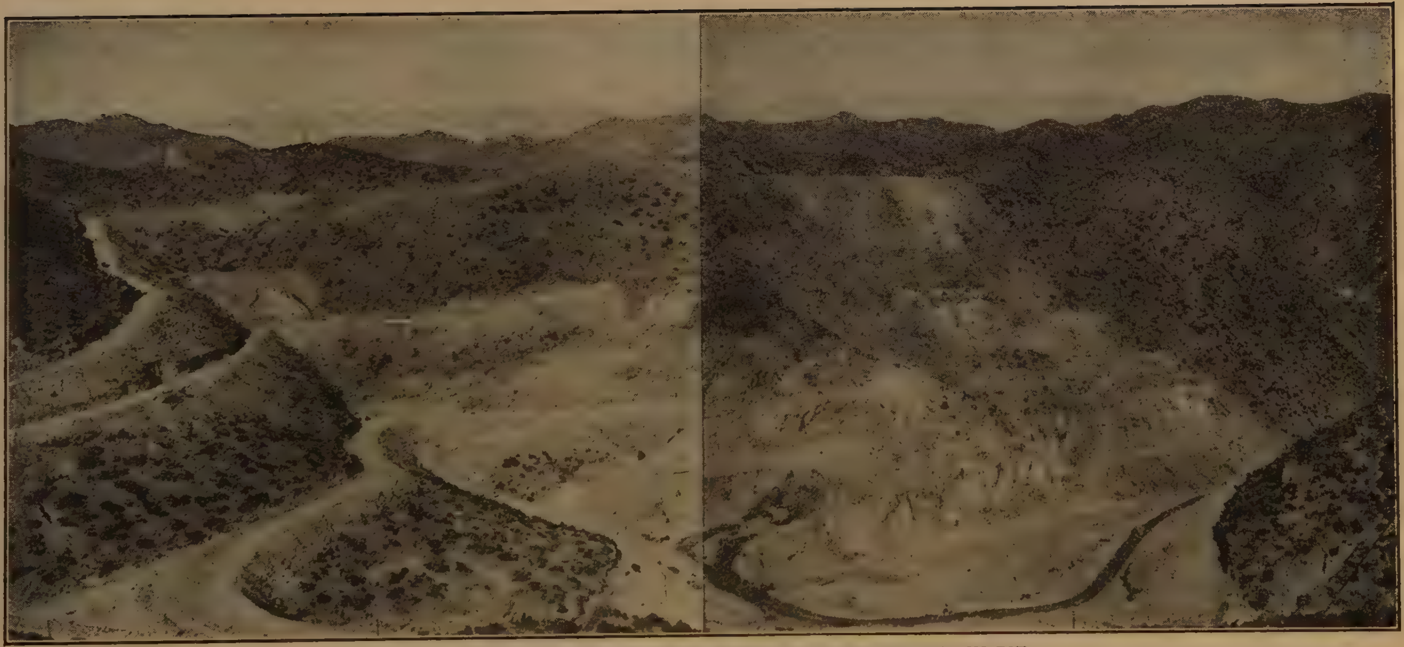


FIG. 3.—ROADS AND BEGINNING OF BENCH DEVELOPMENT IN PIT.

under ordinary boring methods, the contract entered into specified that water loss could not exceed 10 per cent.

The amount of drilling water recovered in turn determines the amount of cuttings recovered. The importance of recovering as much of the cuttings as is practical can be appreciated when it is realized that 100 per cent core recovery from a hole of nominal size represents only about one-third of the total volume of the bore. Thus, when core recovery drops to, say, 20 or 25 per cent, the core itself is a very small proportion of the total hole and it alone probably is not at all representative of the sample interval.

The control of sampling was at all times under the direction of a company engineer, who accurately and carefully measured the amount of water put into a hole for each sample run and compared it with the amount of water recovered. When the water loss came up to the specified limit, drilling was suspended until the hole was put into sampling condition by cementing.

Construction Work

Of necessity, the first preparatory work was the building of roads into the area to be stripped. The nearest automobile road was 1½ miles from the point where the No. 1 shovel would start its operations. Six miles of service and truck haulage roads were made ready within the pit area before the first big shovel was delivered, assembled and moved to the mine. About 75 per cent of all roads constructed were made by drilling with large jackhammers suspended with block and tackle from tripods, blasting the drill holes and grading off with bulldozers. Some few thoroughcuts were made with a small Diesel shovel and 5-yd. trucks.

The mine area was quite inaccessible to rail, the terminus of the Corporation's industrial railroad being about one mile from the pit entrance. The connection between the railroad and the mine was the steep, extremely narrow main street of Morenci, called Burro Alley—a picturesque reminder of the earliest means of transportation in the history of the district. In order that the shovels and other large equipment might negotiate this thoroughfare, two dozen buildings were razed, the ground was drilled, blasted and brought to ample width and proper grade by a Diesel shovel and 5-yd. trucks.

Two existing buildings near the mine, which were suitably constructed and properly located, were repaired and equipped for use as a drill-tool sharpening shop and truck garage. Water lines were laid so as to serve the diamond drills and later the churn drills. Air and electric current were available from an existing plant. Air lines were extended to serve the jackhammers. A power line was built to the edge of the proposed pit and a substation erected there for distribution to the shovels and churn drills.

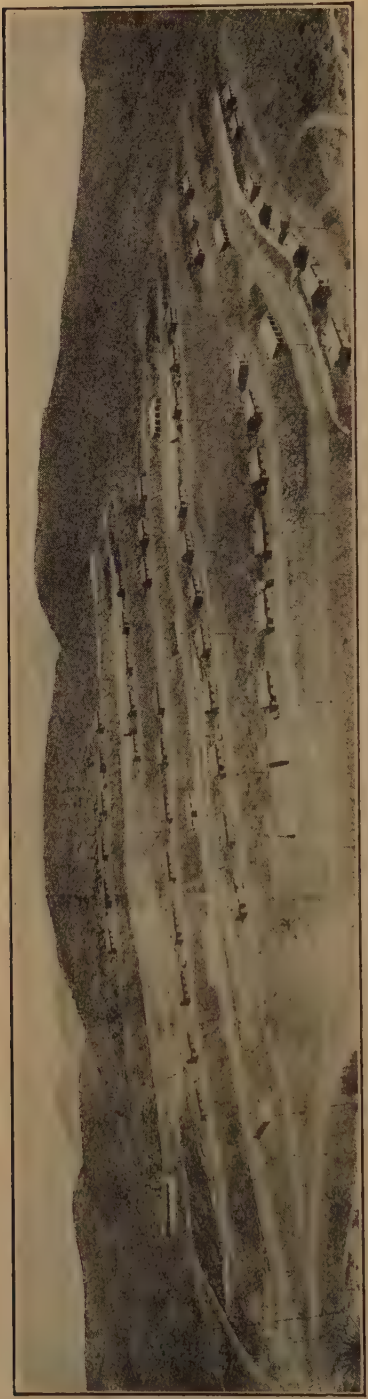


FIG. 4.—NEW TOWNSITE. FIVE-ROOM HOUSES WITH COPPER ROOFS AND COPPER PIPING.

For more than 30 years a 36-in. narrow-gauge railroad adequately served the Morenci mining district. Connection with the Southern Pacific standard-gauge tracks was made at Clifton, 10 miles distant. However, the initiation of open-pit mining and use of the heavy equipment incident thereto, together with the proposed construction and subsequent operation of a large reduction plant, made it necessary to standardize the tracks from Clifton to Morenci. This had to be done before the railroad cars with shovel parts could be brought into Morenci, and required widening of all cuts and fills and replacement throughout with new ties. The excavation was handled with 1-yd. shovels, 5-yd. trucks and bulldozers.

Along with this direct preparatory construction work, a number of indirect yet vital phases of the entire project were under way. Existing houses were repaired; a rooming house, a restaurant and a lounge room for employees were renovated; a staff house for engineers and office workers was rehabilitated and made ready for use; an apartment court for the community school teachers was constructed; club rooms and apartments for Mexican employees went up; and grading was started for the erection of 50 five-room houses in a new townsite (Fig. 4) about midway between the mine and the proposed site of the new reduction plant.

All of the several preliminary engineering and preliminary construction projects were under way concurrently, each being retarded or brought to fruition according to the part it played in the whole program.

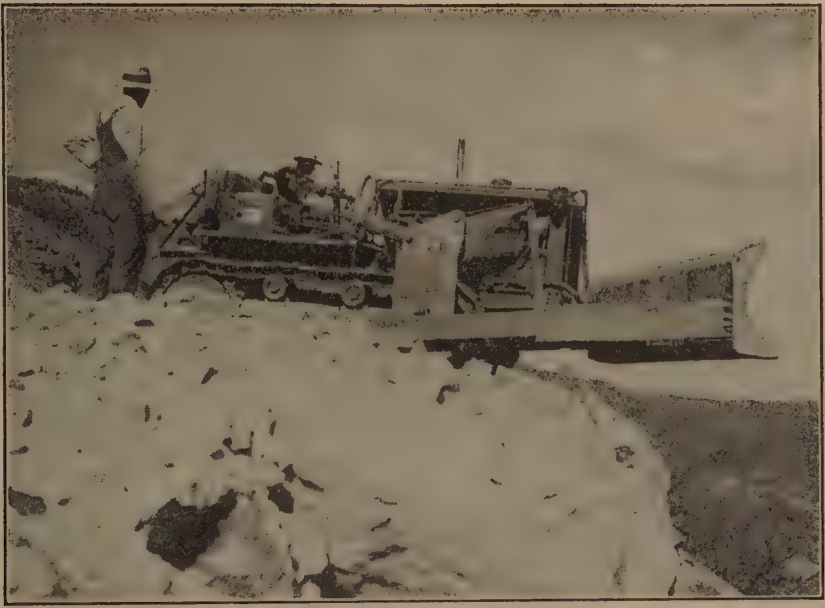


FIG. 5.—DIESEL TRACTOR, 90-HORSEPOWER, EQUIPPED WITH HYDRAULIC BULLDOZER, TOWING WINCH AND LIGHTS FOR NIGHT WORK.

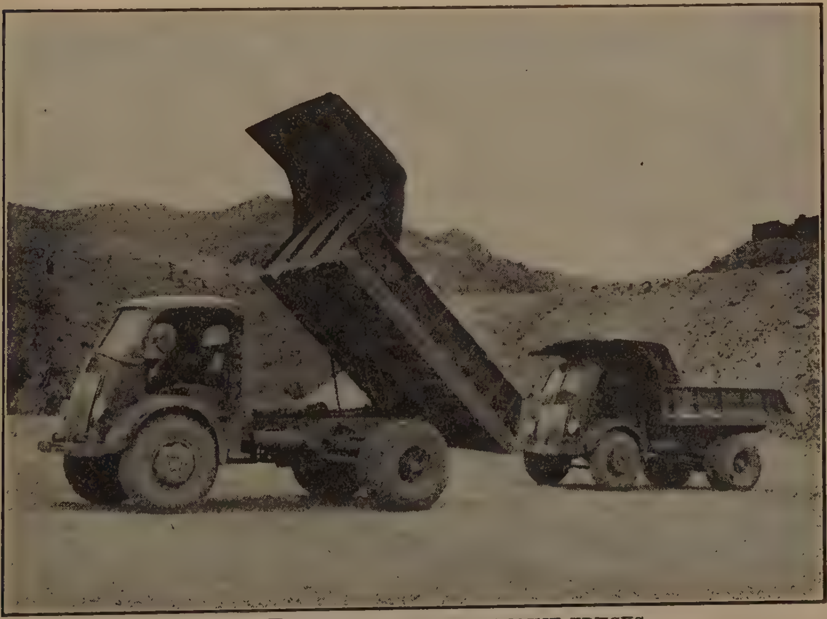


FIG. 6.—FIVE-YARD CAB-OVER-ENGINE TRUCKS.
End dump; gasoline engines; wheel base, 121 inches.

An illustrative example of how well the work progressed with the scheduling is that the last spike was driven in the reconstructed railroad the day that the first carload of big shovel parts arrived at Clifton.

CONSTRUCTION EQUIPMENT

Foremost on the list of construction equipment was jackhammers; next in order, bulldozers.

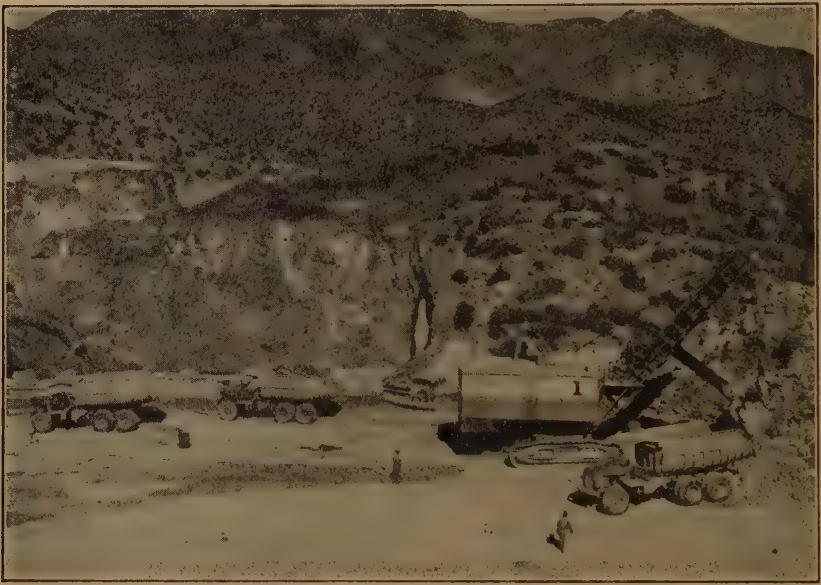


FIG. 7.—ELECTRIC SHOVEL, 4½-YARD, LOADING INTO 22½-YARD TRUCKS. Note trucks on both sides of shovel and bulldozer keeping roadway clear.

Sinker drills chucked for 1¼-in. round steel were used suspended from tripods. Complete converting equipment and regrinding tools were installed in the drill shop to service the drill bits and rods.

Five of the largest type Diesel tractors, each equipped with a hydraulic bulldozer and towing winch, were purchased. These have been in continuous use since. (Fig. 5.)

Two 1-yd. capacity Diesel shovels and five 5-yd. end-dump trucks (Fig. 6) were bought for use in general excavation work. A crane attachment to be used in the erection of the 4½-yd. electric shovels (Fig. 7) was specified for interchanging with the short boom and digging dipper on one of the Diesel shovels. An interesting feature regarding the 5-yd. trucks selected is that they are cab-over-engine type, which allows a wheelbase of 121 in.—only 9 in. greater than on a Ford passenger car. The short cutting radius thereby permitted was of great advantage in tight places.

Several small 1½-ton trucks were needed in moving materials, supplies and employees to and from the various jobs.

SHOVELS

Four electric shovels have been erected and are in the mine ready for use. These are full-revolving, Ward-Leonard control, rated at 4½-yd. capacity. Electric current is delivered to the shovels through a trail cable at 2300 volts potential. Three of the shovels are equipped with 4½-yd., cast-manganese dippers having renewable manganese lips. The fourth dipper is 5½-yd. capacity, fabricated of alloy steel with a lip of cast manganese. The total weight of the 5½-yd. dipper plus its load when fully loaded is somewhat less than the fully loaded 4½-yd. manganese dipper.

Shovel production has been maintained at about 5000 tons per 8-hr. shovel shift during the current year. The entire production has been hauled away in 22½-yd. trucks. Rail and dump cars are not scheduled to be brought into the mine until early in 1940. Self-dumping, drop-door, 30-yd. capacity rail cars will be used for waste. Complete electrification of the haulage system is now contemplated.

The shovels have operated nearly 92 per cent of the possible time. The 8 per cent delay time includes all delays that occur, the major items being waiting for trucks, shovel mechanical repairs and shovel electrical repairs.

During the first five months of this year, the 8-hr. shift cost of operating the electric shovels, including maintenance, was $52.81. The crew consists of one operator, one oiler and two pitmen. One of the pitmen sprays the bank ahead of the shovel with water to keep the dust down. Power consumption averages 0.186 kw-hr. per ton loaded.

Stripping operations have been confined to the operation of two shovels, two shifts a day.

HAULAGE

Eighteen 22½-yd. end-dump trucks (Fig. 8) constitute the haulage fleet. On the Morenci job the trucks haul a pay load of 33 tons. All are driven with gasoline engines. A vital and interesting thing pertaining to the use of trucks is their production capacity and their cost of operation.

As to production, it may be said that trucks make a very flexible operation and quite satisfactorily service the shovels. Bench widths are maintained so a truck may be spotted on each side of a shovel, which greatly reduces the arc of shovel swing and allows an empty to be backed into loading position while the shovel is still busy with the truck on the opposite side.

As to costs, the base line of Morenci operations is not long enough to guarantee that there will be no change. For the first five months of the current year the cost per truck shift was $31, including all operating expense, such as driver's wages, gasoline, oil, tires, truck repairs and road maintenance, but not road construction. The trucks were practically new at the beginning of this period, having been in service on an average of about three months.

The cost per ton of material hauled has been 3.8¢; the cost per ton-mile 6.1¢. The proportionate part of the total cost that the various items bear may be summarized as follows: maintenance, 32 per cent; tires, 21; driver, 19; gasoline, 14; road maintenance, 10; all other, 4; all, 100.

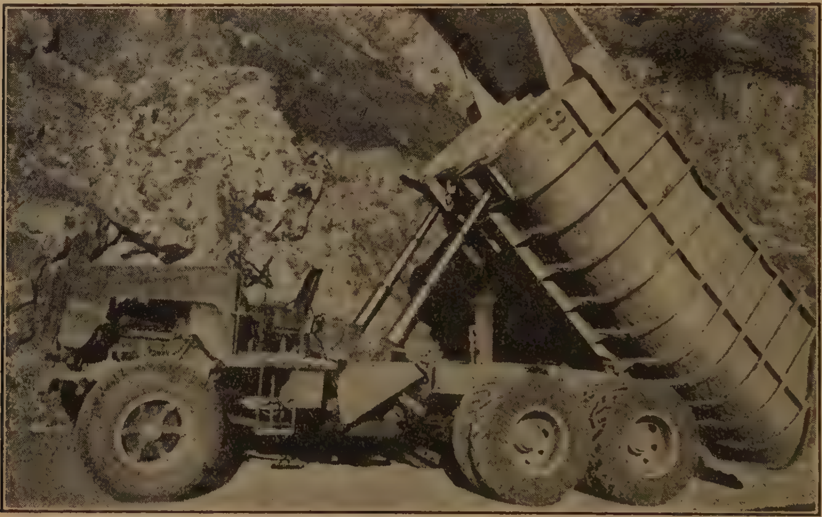


FIG. 8.—END-DUMP TRUCK, 22½-YARD, WITH TWIN TELESCOPIC HOISTS. Chain drive; gasoline engine, 580-cu. in. displacement; low gear ratio, 173 to 1; 13.50 by 24-inch tires in rear.

The average haul has been 0.6 miles; the maximum, about one mile. When the length of haul one way is one mile and a shovel is producing 5000 tons in 8 hr., 10 units are needed in actual operation for uninterrupted shovel service. The top speed on slightly favorable or level grades, either loaded or empty, is 15 miles per hour. Adverse grades reduce speeds severely.

On roads favorable to the loads, grades as high as 15 per cent, at least, can be negotiated; but the trucks must proceed over gradients as steep as this very slowly and if it is at all possible 10 per cent is set as the maximum. For long stretches, a 6 or 7 per cent grade is much preferred. Hauling loads uphill—that is, against the grade—has not been necessary at Morenci.

The trucks have pneumatic tires all around. The cost of one tire is about $200, tubes $25 additional. A complete set of tires for one truck represents $2000 worth of rubber. The average life of tires is about 9000 miles. The average on gasoline is about 0.8 miles per gallon.

While the mining operations are suspended on graveyard shift, the trucks are brought into the main repair shop for servicing. One-half the fleet is serviced every night, so that each truck comes in after 32 hr. of operation. The chassis are lubricated on this interval and the engine oil changed after 64 hr. of operation. Card records are maintained of all shop servicing, repairs and tire changes. A complete tire-repair shop, including vulcanizing equipment, has been installed.

One bulldozer is assigned to each operating shovel to work between it and the dump. At the shovel the bulldozer keeps the pit swept up so that the trucks can readily back into loading position. Ordinarily not much bulldozer work is required on the dumps as the trucks generally spill their load over the slope, but as the dumps settle they are brought up to grade by dumping on the top. At those times, the bulldozers are used to grade down and make a smooth running surface.

Bulldozers are also used very extensively in the drilling and blasting operations. The first churn-drill roads are built on the original hillsides by drilling and shooting a line of 8-ft. air-drill holes and then grading off. Some new road work is also frequently necessary and generally requires bulldozing. The cost per 8-hr. shift of bulldozer operation has been $22.32.

Gasoline is delivered to the trucks and Diesel fuel oil to the bulldozer tractors and small shovels, by a specially designed service truck. In addition to the carrying tank, which is divided into two compartments, for gasoline and fuel oil, complete greasing equipment and a small gasoline-driven air compressor are permanently attached to the truck bed. Although it is the policy to bring the trucks into the shop for greasing and servicing, frequently in emergencies the portable outfit saves much time.

Two sprinkler trucks are operated over the haulage roads. These are very effective in controlling the dust.

Drilling and Blasting

The initial drilling on hillside slopes is being done with jackhammers chucked for 1¼-in. round steel. For plugging boulders smaller machines chucked for 1-in. round steel are used. All drilling in the preliminary operations has been dry. Detachable bits have been used exclusively.

For all primary drilling electrically operated churn drills (Fig. 9) using 9-in. diameter drill bits are standard. Holes are drilled from 8 to 10 ft. below grade.

The porphyry rock offers good drilling conditions, the average advance per drill shift being 85 ft. Drill bits average 55 ft. per use. The bits are reconditioned by a mechanical sharpener. Three men in the sharpening shop—one sharpener, one temperer and one helper—are able to resharpen and harden 22 bits per 8-hr. shift.

The cost per shift of operating a churn drill has been $30.75. This includes the dressing of bits, as this operation is necessarily part of the cost of drilling. The cost per foot of churn-drill hole has been 36¢; the cost per foot of primary air-drill hole has been 14.7 cents.

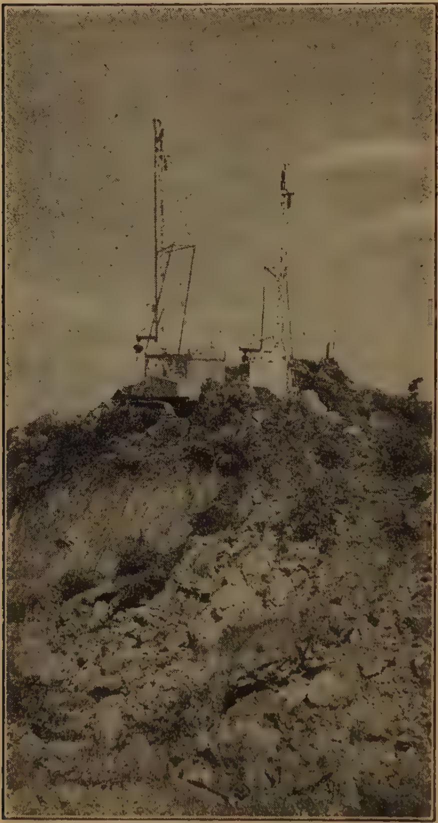


Fig. 9.—Electrically driven churn drills.

Bits 9-in. diameter; weight of tool string, 3000 pounds.

Churn-drill holes are not sprung before loading with explosives. The main powder charge is put in the bottom of the hole and one or two deck loads nearer the top. The principal explosive used is a free-running bag powder. About one case of gelatin is loaded into each hole to serve as a primer.

The detonation of all churn-drill holes is with Cordeau fuse. Where small service trucks can be driven to the vicinity of the holes, old mill tailings are brought in for stemming. With this type of stemming a single line of wire-bound Cordeau is used in the holes. Where the drilling is inaccessible to service trucks, stemming is made by one or more screenings of the material at hand near the holes. In these places, where the coarser stemming offers a greater hazard of collapsing the fuse, a double priming is provided by putting two lines of double-countered Cordeau in each hole.

Deep air-drill holes are sprung before loading. Air-drill holes are sometimes detonated with Cordeau and sometimes with electric caps, the choice depending upon the location of the shot. Secondary plugger holes are fired with ordinary fire fuse and caps generally, although at times a number are fired at one time with electric caps.

The character of the capping so far has been variable, therefore the degree of fragmentation has varied from fine to coarse. In general, the outermost shell seems to be hardest and breaks largest. In places, ribs of material high in quartz are found, which are difficult to break satisfactorily. The trend so far indicated is that the material becomes progressively softer with depth, and it is believed that the ore will give good fragmentation throughout.

THE OUTLOOK

As is true of most copper mines, it was gold, not copper, that brought the first miners into the Morenci district. The search for gold was abandoned for the production of copper nearly 70 years ago. It seems probable that the present development will be good for at least 30 years more—the new mine added to the old mines is going to round off a century of production.

DISCUSSION

(*William Koerner presiding*)

E. M. J. ALENIUS,* Morenci, Ariz.—Diesel-powered trucks were not installed at the Morenci open pit because no manufacturers were prepared at the time to furnish trucks suitable for the intermittent short-haul operations and the grades upon which the trucks would be required to descend. The gasoline engine is better for frequent starting and stopping, for short hauls, and for shifting gears on the various grades encountered. Supplementary braking equipment would be needed on Diesel-powered trucks to provide effective control in hauling heavy loads down a steep grade, as the Diesel engine does not assist as does the gasoline engine, by the use of compression.

One type of truck is equipped with an engine with a rating of 130 brake-horsepower at 2050 r.p.m.; the engine on the other type is rated at 150 brake horsepower at 1800 r.p.m. The ¾-yd. and 1-yd. shovels are powered with a Diesel engine having a continuous rating of 60 b. hp. or a maximum rating of 80 b. hp. at 850 revolutions per minute.

---

* Pit Foreman, Morenci Branch, Phelps Dodge Corporation.

Recent Improvements in the Mining Practice of the Tri-State District

By C. W. Nicolson,* Member A.I.M.E.

(New York Meeting, February, 1938)

The Tri-State zinc and lead-mining district is in the northeast corner of Oklahoma, the southeast corner of Kansas and the southwest corner of Missouri. The area throughout which active mining has been conducted is about 45 miles long and about 5 miles wide, extending northeasterly from Commerce, Okla., to Alba, Missouri.

Although the lead ores were mined on a very small scale in the vicinity of Joplin, Mo., as early as 1848, nothing of any importance was done in the district until 1870, since when both mining and smelting have been prosecuted with considerable vigor. In the beginning the lead ores only were exploited, but within a few years extensive deposits of zinc ores were found and since that time the production of zinc ore has far exceeded that of lead.

Both the surface and mineral rights in Missouri and Kansas were under individual ownership by the time ore was discovered. Although some of these rights were purchased by mining companies, most of the mining has been done under leases covering small areas, usually 40 acres, although in the past many leases were given on much smaller tracts. In Oklahoma most of the mineral land had been alloted by the Government to various Indian tribes and although the title to some of these lands had been acquired by various white men before the discovery of the rich Picher field, the title to most of the land has remained with the Indian tribes and the leasing of the mineral rights has been done by the Federal Government for the benefit of the Indians. The Indian allotments were of 200 acres each and some of the mining leases covered entire allotments, although many leases were granted for only 40 acres or even less.

Practically all of the leases granted in the past required the erection of a concentrating plant on each lease, a condition that has had a pronounced effect upon the mining and milling methods of the district, since the apparent ore reserves on each tract were too small to justify large and elaborate mining and milling plants. Most of the leases were

---

Manuscript received at the office of the Institute Nov. 22, 1937. Issued as T.P. 905 in Mining Technology, May 1938.

* General Superintendent, Eagle-Picher Mining and Smelting Co., Picher, Okla.

acquired by partnerships or by small corporations; few, if any, of which were sufficiently well financed to conduct a campaign of exploration that would prove the true limits and values of the ore deposits. As a result, many of the original concentrating plants were small and inefficient and many of the mining methods adopted were those requiring the least original expenditure. Furthermore, the large and comparatively rapid fluctuations in metal prices throughout the life of this district have produced alternating booms and depressions, which have discouraged development of a long-range policy and tended to prolong the "hand to mouth" policies of the smaller producers.

In recent years, however, many of the leases have been acquired by large mining companies and mining and particularly the milling methods have been greatly improved.

The discovery of the rich district near Picher, Okla., in 1914 and 1915 produced a tremendous boom, since at that time, owing to the war in Europe, the metal prices were very high. The ore deposits in this district are very irregular and difficult to outline by surface exploration. The natural desire for quick profits led to the opening of mines and the erection of mills with the least possible amount of exploration. This resulted, as before, in small, inefficient units and the adoption of mining methods that although cheap to inaugurate, proved expensive over a period of time.

Geology

The geology of the district has been described by Fowler and Lyden[1] and no attempt will be made here to deal with it in detail. It may be said, however, that the rocks of the district are sedimentary, dipping gently to the west, and that the ore deposits occur in certain strata that have a maximum depth of about 150 ft. at the east edge of the district and of some 400 ft. at the west edge. Although these strata when deposited were limestone, many of them, particularly the ore-bearing beds, have been silicified and have become flint and chert.

The ore deposits are of two general types: (1) the so-called sheet-ground deposits, which are of considerable lateral extent but usually only 8 to 10 ft. in depth, and which occur in the lowest of the mineralized beds, particularly in the Missouri district, and (2) the brecciated and stratified deposits in the upper beds, particularly in the Picher district, in which the thickness of the ore deposits ranges from 10 to over 100 ft. and which, although of considerable length, are usually relatively narrow.

The average metal content of the ore of this district is low as compared with that of many other districts. With normal metal prices the average ore has a value of about $2.50 per ton, a condition that has, of course, naturally affected the mining methods employed. The fact that the gangue is an extremely abrasive flint also had a pronounced effect on the mining methods employed and on the cost of production.

---

[1] G. M. Fowler and J. P. Lyden: Ore Deposits of the Tri-State District. *Trans.* A.I.M.E. (1932) **102**, 206.

For the last 20 years most of the mining has been in the breccia deposits of the Picher district, which are richer than the sheet-ground deposits formerly worked. However, during the past year some of these sheet-ground deposits have been reopened and it is probable that in the future production from the Missouri district will increase.

Exploration

Exploration in the district has been almost exclusively by churn drilling, because of the shallow depths of the ore horizons. A few diamond-drill holes have been put down, but as the rocks, particularly in the Picher district, contain many vugs and cavities, and as the mineralization is very spotty and irregular, they have proved to be expensive and unreliable. Churn-drilling is comparatively fast and inexpensive, the average depth sunk being about 4 ft. per hour and the average cost about $1 per foot. The standard rig for the district is the No. 5 Keystone, using a 6¼-in. bit. The cuttings are bailed every 5 ft. in the overlying rocks and every 2½ ft. in the ore horizons. On account of the irregularity of the ore deposits, there can be no fixed plan for exploration. Holes are usually sunk at intervals of 200 to 400 ft. until ore is discovered, after which they are spaced at intervals of 50 ft. or more, as seems to be justified in each instance.

Formerly all drill rigs used 2⅛-in. manilla rope for the tools, but recently most of the rigs have been adapted to use ⅞-in. 6 by 19 wire rope by placing coiled springs under the walking-beam sheave, and sometimes under the crown sheave. These changes have reduced the cable cost by about 66 per cent without affecting the drilling speed materially. The mobility of the rigs has been greatly increased by mounting them on truck chassis, using the truck engine for power, both in moving and drilling.

The methods of exploration and of mining were described in the U. S. Bureau of Mines *Inf. Circ.* 6113 and *Inf. Circ.* 6121, published in April and May of 1929. No attempt will be made here to describe them in detail, the purpose of this paper being to describe the newer methods that have been introduced and the reasons for such changes.

Drilling and Blasting

The equipment used in different mines varies according to the nature and hardness of the rock at the different properties. However, all drilling, except for blockholing, is done by drifters with cylinder diameters of 3 in., 3½ in. and 4 in. They are mounted on columns and arms in the headings and haulage drifts, and on tripods on the benches. The air pressure varies from 80 to 90 lb. at the machine and all drilling is done wet with hollow drill steel.

Although various weights and cross sections of steel have been used, most companies have now standardized on 1¼-in. round hollow steel. Drill steel and bits have always been a very important factor in mining costs, since the flint is hard and extremely abrasive. Prior to 1935, drilling was done almost entirely with forged bits, using a starting bit with a diameter of 2⅝ or 2¾ in. and with a drop in gauge of ¼ in. for each 2-ft. run. In some of the mines in which the rock was slightly less abrasive a drop of ⅛ in. was used. Since most of the holes in the headings are 8 to 10 ft. deep, and in the benches 16 ft., the finishing bits were usually 1½ or 1⅝ in. Because of the loss in gauge in the last bit, the bottom of the hole was frequently little larger than the diameter of the drill steel itself. In some of the less abrasive rocks a single bit could be used twice and occasionally three times, but in the more abrasive rocks a bit could be used only once, because of the excessive loss of gauge.

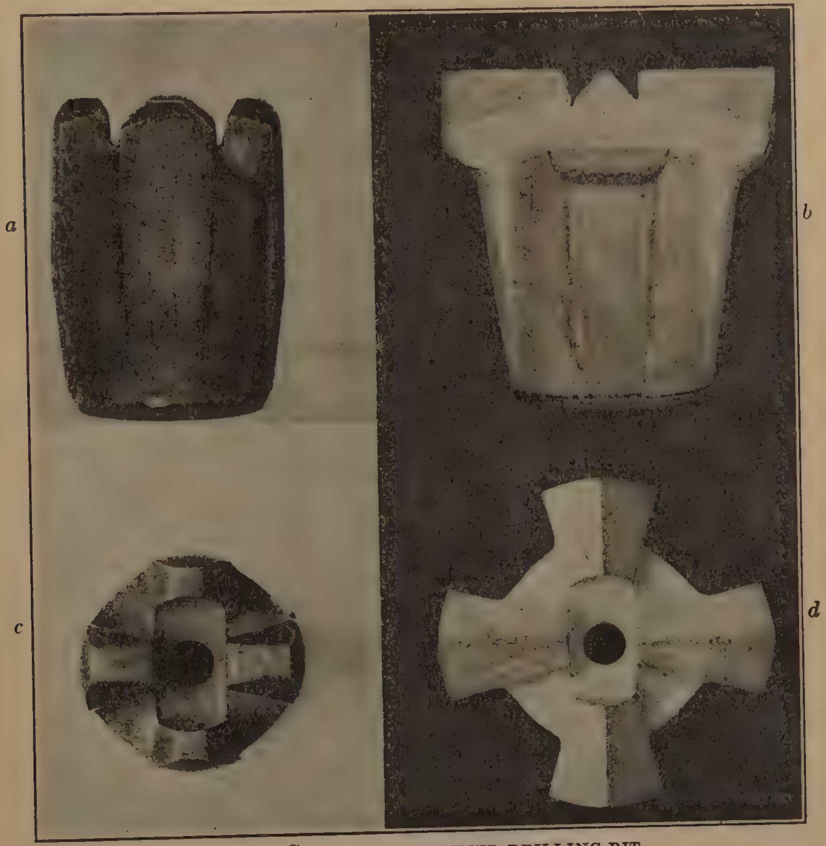


FIG. 1.—SPECIALLY DESIGNED DRILLING BIT.

The average distance drilled per machine shift is 60 ft., although in heading ground it is not unusual to drill 100 ft. or even more. With forged bits this frequently meant the dulling of 50 or more pieces of steel by each machine per shift. Although the handling of steel underground in these mines is not as difficult and expensive as in mines with many long raises, nevertheless transportation of 300 to 500 pieces of steel per shift involved considerable expense and delay in hoisting. Detachable bits of various designs had been tried occasionally in different mines, but had never been successful because loss of gauge was so rapid that usually only one regrind was possible after the bit had been run 2 feet.

A study of the mechanics of drilling in flint showed that the rock was not cut by the sharp edge of the bit, but that pieces of rock were flaked off by the impact of the blow, and that with the ordinary four-point cross-bit these flakes were pushed around the periphery of the hole until ground almost to powder, and that it was this grinding that caused the rapid abrasion. A bit was designed, therefore, as shown on Fig. 1, in which the four individual wings were separated so as to permit the large flakes to be washed out of the hole immediately without moving out to the periphery.

As it had been noted that the edges of a new bit were dulled during the first few inches of the run, it was realized that sharpness was not a necessary condition and that drilling could be done as well and as rapidly with a dull bit. For this reason it was determined not to attempt to regrind the bits, but to use each bit continuously until the wing diameter had been reduced to the skirt diameter. To do this each bit is changed from one rod to the next and used until the hole is completed. In the less abrasive rocks of the district the same bit can be used for one or more additional holes, so that some bits drill as much as 60 ft. without regrinding or other attention. However, in the most abrasive rocks the life of the bit as originally designed was about 12 ft., which limited holes to that depth or required the use of forged bits to bottom the hole.

Only one size of the bit was manufactured, which as originally designed and as used extensively in the district during the past 18 months, was 2½ in. in diameter with a skirt diameter of 1⅝ in. However, this experience has recently led to the adoption of a 2⅝-in. bit, which is now being used. This bit adds about 15 per cent to the footage that can be obtained from a bit and obviates almost entirely the necessity for the use of small forged finishing bits. In the original bit (Fig. 1) small supporting ribs were formed under the wings. It was thought that they were necessary to prevent the wings from breaking off, but in the latest design they have been omitted, since their support proved to be unnecessary, and by protruding beyond the skirt diameter they rendered rotation difficult during the last 2 or 3 ft. of drilling. Table 1 shows the results obtained with this 2½-in. detachable bit at several mines in the district as compared with the average of forged bits. Table 2 shows the comparative cost of drilling with forged bits, with the ordinary type of detachable bit and with the A-2 detachable bit, which has just been described.

TABLE 1.—*Results with Detachable Bits and Forged Bits*

Mine	Holes Drilled	Feet Drilled	Feet per Bit	Bits Dulled	Steel Broken	Feet per Break
DETACHABLE BITS						
Admiralty	8,319	76,020	20.0	3,809	669	113.6
Beck	1,770	15,199	21.6	703	123	123.6
Big John	7,017	55,724	24.4	2,281	860	64.8
Black Eagle	4,090	41,562	11.8	3,529	316	131.5
Coon Dog	1,195	10,823	13.9	781	87	124.4
Ohimo	4,854	43,293	17.6	2,459	127	340.9
Tulsa	2,768	23,481	25.9	906	148	158.6
D. C. & E.	2,999	26,785	16.1	1,668	181	148.0
Total	33,012	292,887	18.15	16,136	2,511	116.6
FORGED BITS						
Mines listed above	4,373	36,644	4.3	8,570	676	54.2
Other mines	33,265	427,695	3.43	124,367		

TABLE 2.—*Comparative Costs*

	Forged Bits		Standard Detachable Bits	A-2 Detachable Bits
	Per Bit Dulled	Per Foot Drilled	Per Foot Drilled	Per Foot Drilled
Sharpening:				
Labor	$0.07027			
Supplies	0.01035			
Fuel	0.00752			
Compressed air	0.01622			
	0.10436	0.0304		
New bits			0.0288	0.0172
Re-treatment			0.0070	0.0016
Rods, threading			0.0020	0.0020
Steel loss	0.03000	0.0087	0.0052	0.0062
Transportation and nipping	0.01600	0.0047	0.0020	0.0020
Interest on steel inventory	0.01720	0.0050	0.0006	0.0006
Maintenance of shop	0.00195	0.0006	0.0005	0.0005
Total	0.16951	0.0494	0.0471	0.0301

All rods used with these detachable bits are of 1¼-in. hollow round drill steel and have 1¼-in. threads. The rods are from the stock of drill steel that was in service, supplemented by some new steel. For some time the threads were forged on the rods by special dies in a sharpening machine, but at the present time all threads are being cut by tools in a threading machine. It was found difficult to maintain the proper diameter of the thread with the dies and although the forging of the threads was much faster and cheaper this was offset by the fact that the cut threads are more nearly perfect and that the total number required per day can be turned out by one man at the rate of eight per hour.

The thread end of the rod is heated to about 1800° and enlarged by punching, after which it is annealed in lime. The thread and button are then cut, and the end of the rod to a length of about 5 in. is heated to 1520° and quenched in oil. The shanks and lugs are forged at 1800°, cooled in air, reheated to 1500° to a length of about 10 in., and quenched in oil. The heating forges were designed and made in the mine shops; they use natural gas as a fuel with low-pressure air. The reheating forge is of the muffle type, using natural gas and low-pressure air, the temperature being indicated by an electric pyrometer.

It has been noted that the breakage of rods per foot drilled is less with these detachable bits than with forged bits. Table 3 shows the results over a period of months with rods for detachable bits made from a stock of new Swedish steel, the same rods being in service throughout the entire period.

TABLE 3.—*Results with Swedish Steel*

Month	Holes Drilled	Feet Drilled	Bits Used	Feet per Bit	Pieces Steel Broken	Feet Drilled per Broken Steel
March	1,082	8,913	431	21.17	10	891.3
April	984	9,193	522	17.61	23	399.7
May	729	6,419	407	15.80	29	221.4
June	677	6,273	445	14.10	21	298.2
July	533	4,505	267	16.90	14	321.8
August	849	7,990	397	20.10	30	266.3

DRIFTING AND STOPING

No attempt will be made here to describe these methods in detail because they are well described in the Bureau of Mines *Information Circulars* 6113 and 6121. However, it may be said briefly that the room-and-pillar method is used, the rooms being advanced with headings and benches. The headings are usually about 8 ft. high and the benches anything from 10 to 100 ft., depending upon the thickness of the ore. Horizontal holes, usually 16 ft. long, are drilled in the benches to take advantage of the stratification of the rock. The tonnage broken per foot of hole drilled and per pound of dynamite used varies greatly in different mines, but the average of the district is about 0.641 tons per foot of hole drilled and 1.3 tons per pound of dynamite.

UNDERGROUND LOADING

The standard method of the district has always been by hand shoveling. From time to time electric shovels have been tried underground, with, on the whole, little success. The irregular outline of the ore deposits, the abrasive nature of the rock, the tendency of the rock to break into large boulders or slabs, the frequently steep grades of the haulage tracks and the limited ore reserves in sight were factors that usually so decreased the efficiency and increased the operating and maintenance costs as to render this method of loading unsatisfactory. Small air-driven loading machines were also tried, but the costs of repairs and maintenance were excessive. The local method of shoveling into cans, while comparatively expensive, had many advantages in the smaller mines of the district, particularly in the long, narrow ore bodies. The flexibility of the method, the small initial expenditures required and the comparatively large tonnages shoveled by each man greatly delayed the adoption of any system of mechanical loading. The average shoveler of the district loads about 25 tons per shift and many of these men under favorable conditions have averaged as high as 50 tons.

During the last year, however, several of the larger companies have installed scrapers. To some extent this method may still be considered as being in the experimental stage, but the results obtained indicate that scraping is successful under the proper conditions and that the number of scrapers in use in the district will be materially increased. Since in most of the mines there are no development drifts below the ore bodies from which raises can be put up, it is usually necessary to load directly into cars or cans, although in a few mines upper levels have been developed from raises. Cars are used in only a few mines and the almost universal system of haulage is in cans mounted on small, low cars. The cans are 32 in. in diameter, require a vertical clearance of 54 in. and hold about 1400 lb. of rock. Being so small, they constitute difficult units into which to load with a scraper. The systems of loading by scrapers, which have been developed so far, may be classified as follows:

1. Scraping into raises or storage hoppers.
2. Scraping into cars or cans.
    - *a.* With permanent wooden ramps.
    - *b.* With semipermanent wooden ramps.
    - *c.* With portable steel slides.
3. Scraping to conveyor belts.

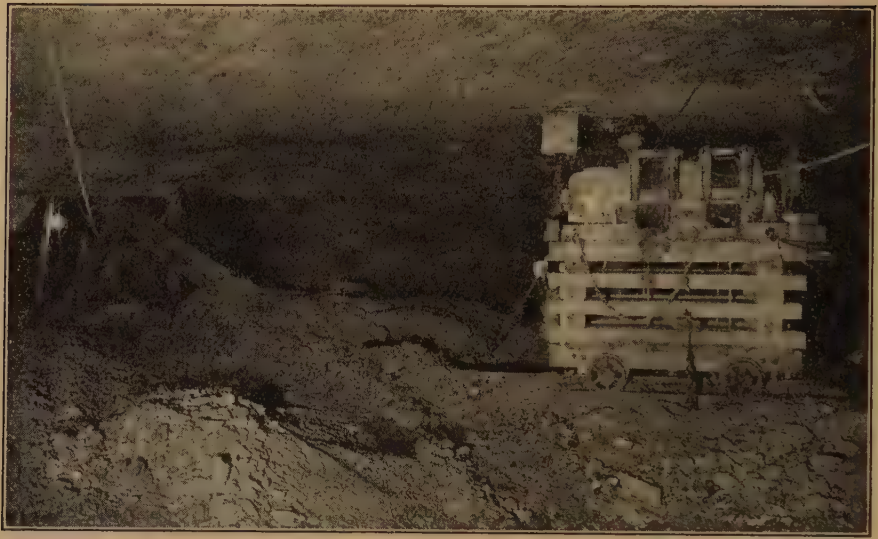


FIG. 2.—SCRAPING TO STORAGE HOPPER ON LOWER TRACK LEVEL.

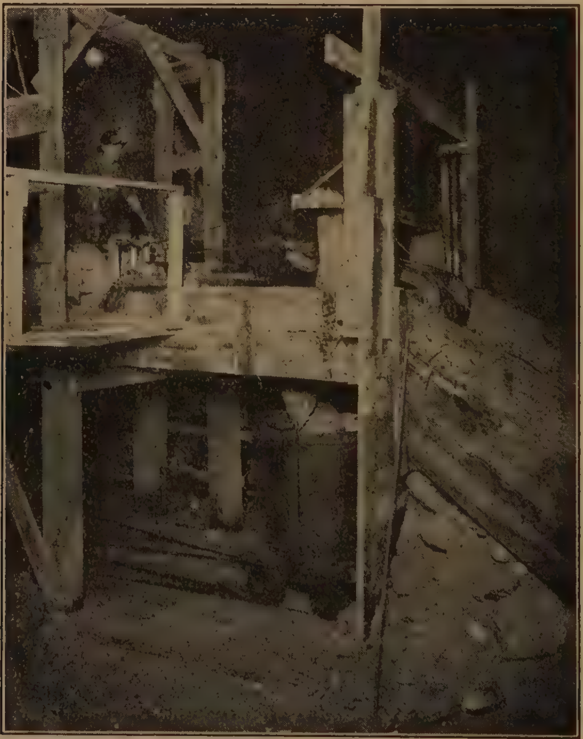


FIG. 3.—PERMANENT WOODEN RAMP WITH CAN UNDERNEATH.

Concerning the method of scraping to raises, there is little, if anything, unusual to be noted. Fig. 2 shows the method of scraping to a storage hopper located on a lower track level, which is in all respects equivalent to scraping to a raise, except that in the installation illustrated here the hoist is mounted on a car that can be moved to different headings, all tributary to the same hopper. Fig. 3 shows a permanent wooden ramp with a loaded can beneath it. In this case each can must be pushed into position, loaded and pushed out again, a slow process that sometimes cannot be avoided. Whenever possible the track passes through the ramp, as in Fig. 4, so that an entire trip of cans can be handled. Fig. 5 shows an installation in which the entire trip of six cans can be pulled endways under the ramp, this trip being pulled into position by the small tugger shown at the right and dropped back by gravity.

FIG. 4.—TRIP OF CANS HANDLED THROUGH RAMP.

The use of the two semipermanent ramps is illustrated in Fig. 6, the hoist mounted on a high car being moved from one ramp to another as desired. These ramps can be dragged forward by the hoist to a new position as the headings are advanced. A portable steel ramp mounted on a turntable supported on a small car is shown in Fig. 7.

Three types of hoists are in use in the district, air-driven two-drum hoists, electric-driven two-drum hoists and electric-driven three-drum

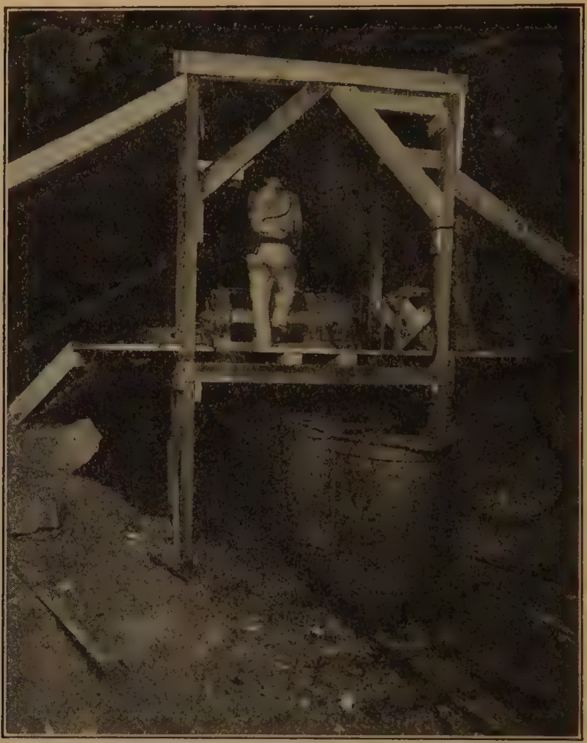


FIG. 5.—TRIP OF SIX CANS PULLED ENDWAYS.

FIG. 6.—SEMIPERMANENT RAMPS.

hoists. In some of the older mines, which have been extensively developed but in which electric service is not available throughout, it has seemed more economical to use air hoists, since air lines had already been installed, rather than to put in the wiring necessary for electric hoists. In the narrower headings, up to 40 ft. in width, two-drum hoists have been satisfactory, but in the wide headings, three-drum hoists have been proved to be advantageous. Fig. 8 is a good illustration of conditions frequently encountered in these mines.

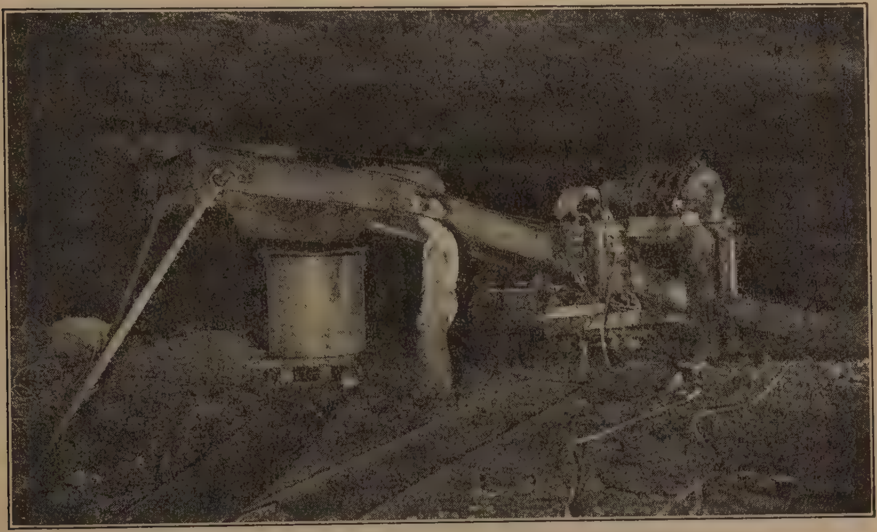


FIG. 7.—PORTABLE STEEL RAMP MOUNTED ON TURNTABLE.

With so many different conditions existing in the district, the costs of scraping vary widely, of course, in different mines. The figures given in Table 4 are averages for all mines, and, as they include the results of experimental installations, are probably higher for scraping than those that eventually will be obtained.

TABLE 4.—*Loading Costs per Ton Delivered on Lay-by*

Operating Data	Scraping to Hopper	Scraping with Permanent Ramp into Cans	Scraping with Portable Ramp into Cans	Scraping with Two Semi-portable Ramps into 2½-ton Cars	Scraping to Conveyor Belts	Hand Shoveling
Tons per shift	144	190	182	123	200	
Tons per man-shift	48	95	61	61	100	25
Scraper operators	1	1	1	1	1	
Scraper helpers	1	1	2	1	1	
Trammers	1	0	0	0	0	
Total cost per ton	0.13	0.09	0.09	0.10	0.08	0.21
Power cost per ton	0.012	0.015	0.015	0.015	0.012	

Most of the scrapers used in the district are made of cast steel with chrome-molybdenum shoes, and have given very good service in spite of the abrasive nature of the rock. Three sizes of scrapers are in use, the 36-in. scraper with air-driven hoists, the 42-in. scraper with air-driven or 15-hp. electric hoists and the 48-in. scraper with 25-hp. electric hoists. No attempt has been made to build up the cutting shoes with hard surfacing materials, the tendency being rather to wear out the shoe and discard it. A 10-in. shoe on a 48-in. scraper has a life of about 11,000 tons.

FIG. 8.—CONDITIONS THAT FREQUENTLY OCCUR IN TRI-STATE DISTRICT.

On the larger scrapers, $\frac{5}{8}$-in. cables are used to pull in and $\frac{1}{2}$-in. or $\frac{9}{16}$-in. to pull back. Because of the abrasive nature of the rock, the life of cables is comparatively short. The present tendency is to use seale-lay cables with wire cores, even though they are less flexible than other kinds.

TRAMMING AND HAULING

From the early days of the district, the almost universal method of haulage has been in cans resting on small flat-topped cars. With a few exceptions, this system was continued in the larger mines of the Picher area, the only change being in the increased size of the cans. The cans, which resemble sinking buckets, are 30 or 32 in. in diameter and 30 in. high. The small mines use 30 by 30-in. cans, which hold 1100 lb. of ore, while the larger mines use 32 by 30-in. cans holding 1200 lb. and 32 by 32-in., holding 1400 lb. The gauge of the track varies from 14 in. in the smaller mines to 24 in. in the larger ones, 8-lb. and 12-lb. rails being used for mule haulage and 20-lb. to 35-lb. rails for mechanical haulage. Many of the mines still use mules, supplemented by small electric or air-driven hoists for handling the trips on the inclines. Storage-battery locomotives and trolley locomotives are used on the main lines in several of the larger mines and in one mine in Missouri gasoline locomotives are used underground.

Since the main-line tracks usually follow the grade of the bedrock, which in many mines rolls considerably, the grades both with and against the loads are sometimes excessive. To do away with the use of hoists on these inclines, a system of endless-rope haulage has been installed in several mines, handling the trips over a distance as great as 3000 ft. Table 5 gives some costs of these different methods of haulage, but it must be understood that the conditions in the various mines are not necessarily comparable and that these figures do not include the costs of cuts and fills necessary for a practicable grade if locomotive haulage is used. The endless-rope haulage is very flexible and by the use of auxiliary ropes trips can be handled on tracks branching from the main lines.

In two of the recently equipped mines in the Missouri district, belt conveyors have been installed underground. Scrapers deliver the broken ore to reciprocating feeders or to short feeder belts, which in turn load the main belt. The main belts convey the ore to shaft pockets, from which it is hoisted by skips. Both of the installations are comparatively short—one is 500 ft., the other 700 ft.—but they will be extended from time to time as work progresses. The use of conveyor belts underground is not at all new, but the handling of an abrasive rock such as flint, and in rather coarse sizes, has presented some interesting problems. The scrapers deliver the ore through grizzlys spaced at 10 in. but the rock tends to break in slabs and some rather large pieces occasionally pass through the grizzly, so must be handled by the belt. Since it was desired to handle only about 600 tons per shift in each of the mines, 24-in. belts were purchased. These are driven at a speed of about 200 ft. per minute. As it was anticipated that the duty of handling flint in large pieces would be severe, the belt was made to the following specifications: 6-ply, 42-oz. duck, skim coat between plys, $\frac{3}{16}$-in. top cover, $\frac{1}{16}$-in. pulley cover with breaker strip in the top cover. The belt being stiff, it is necessary to use 36-in. head pulleys and 30-in. tail pulleys. Three-pulley troughing

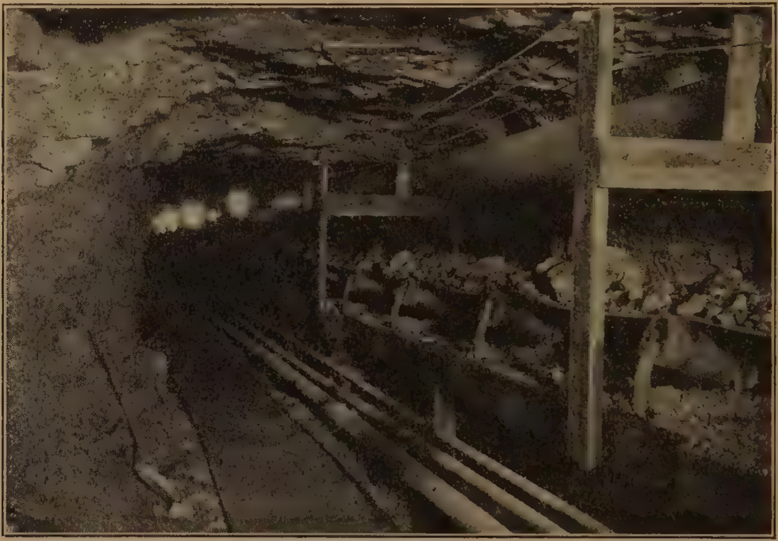


FIG. 9.—LOADED BELT.

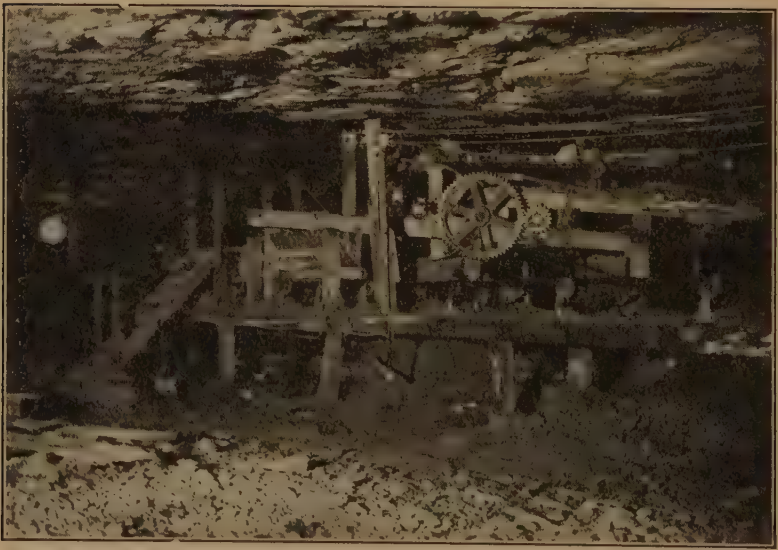


FIG. 10.—TRANSFER POINT BETWEEN BELTS WITH GEAR GUARDS REMOVED.

idlers are used, the pulleys being 4 in. in diameter with sealed bearings, and are placed on 4-ft. centers with return idlers on 16-ft. centers. The belts usually handle about 100 tons per hour, although the capacity is considerably more than this figure. As these belts have been in service only about three months, no idea can be formed as to their ultimate life. Fig. 9 shows the loaded belt and Fig. 10 the transfer point between belts, the gear guards having been removed.

TABLE 5.—*Costs of Haulage*

Type of Haulage	Tons per Man-shift	Labor, Cost per Ton
Animal	116	$0.044
Endless rope	204	0.026
Battery locomotive	240	0.022
Trolley locomotive	260	0.020
Conveyor belt	600	0.009

LIGHTING

All main haulageways are electrically lighted and all rooms in which scrapers are operated are illuminated by flood lights. Most of the workmen of the district use carbide lamps underground, but one large company has standardized on electric cap lamps equipped with special reflectors, which produce strong central beams with a brilliance almost equal to that of a five-cell flashlight. The electric lamps are particularly advantageous in the large rooms in some of the mines and are popular with the workmen. It has always been the custom of the district to provide carbide for the lamps; and the cost of the electric lamp is no more than that of the carbide alone. Including loss from slacking and wastage, carbide costs $0.028 per lamp-shift, while the cost of electric lamps, including amortization of the purchase price, is $0.030 per lamp-shift.

HEADFRAMES

Until recently all headframes and storage hoppers at the shafts were built of wood. The almost universal practice of the district has been to install the hoisting engine in the headframe directly above the shaft. Recently, however, steel headframes and hoppers have been erected. These hoppers hold about 500 tons of rock, which can be drawn off directly into railroad cars for transportation to the central concentrating plant.

SAFETY AND WELFARE

The large companies in the district require all employees to wear safety hats underground, the usual practice being for the company to pay one-half the cost of each hat, the employee paying the other half.

Most companies also require all employees, whether underground or on surface, to wear hard-toed shoes, which usually are purchased by the companies in large quantities and sold to the employees without profit.

The problem of dust in the mine air is one to which a great deal of attention has been devoted for some time. Where ore is loaded by hand shoveling it had been found that satisfactory dust counts could be obtained by wetting the floor and the pile of broken rock two or three times a day. With the introduction of scraping, however, it became necessary to take additional precautions, and various types of sprays were purchased or built. The sprays now in use are made in the mine shops and consist essentially of short lengths of pipe to one end of which a nozzle is attached and into which both water and compressed air are introduced. Although many different types were tried, including those with elaborate mixing chambers, interior injector tubes, and other refinements, it has been found that the simple type is quite as effective as the more elaborate ones. Two sprays are used in each heading, one playing upon the pile of broken rock, the other forming a curtain in front of the scraper operator. This, combined with a liberal washing down of the walls and roof by means of a 2½-in. fire hose, has produced very satisfactory results. Dust counts by the impinger method are made each week and written reports are circulated to all mine foremen, safety inspectors and superintendents.

DISCUSSION

(*J. Wilbur Van Evera presiding*)

G. SHERMAN,* New York, N. Y.—Your use of dull bits is against usual practice. Do you find any particular increase in the breakage of steel or repairs to drilling machines? Dull bits tend to develop a hammer and anvil effect. Do you use different kinds of steel for the bits and rods?

C. W. NICOLSON (author's reply).—As to the breakage of steel, we found a condition that was surprising in that the breakage per foot drilled was less with detachable bits. We had anticipated quite the opposite result, because of the more frequent use of each rod. It may be that the almost constant movement of forged bit steel into and out of the mine and shop was more harmful than we had realized, producing small nicks and cracks in the steel, which weakened it, and it is probably true that recently the heat-treatment, particularly of the shanks, has been more carefully done and more closely supervised.

As to repairs for the drills, it is difficult to make an exact comparison with previous experience, as most of the machines in use now are new, whereas in the past few years we had managed with old machines. The machine repairs normally required in that district are probably greater than in many other districts because of the condition of the rock, which is badly shattered and contains many vugs and fissures. Therefore the steel frequently runs off-line and makes rotation difficult. As an example, rifle bars as hardened for use in other districts fail frequently and some manufacturers are now using a special heat-treatment for the Tri-State district. Briefly, we feel that the machine repair cost now is little, if any, more than it was before and certainly not enough to offset the other advantages we have obtained. Any increase in repair cost is probably due more to difficult rotation than to any hammer-and-anvil effect—that is, the increased wear or breakage is on the rotative parts, such as the rifle bar, rather than on the piston.

---

* Consulting Engineer.

The steel used in the rods is standard drill steel—about 0.85 carbon—whereas for the bits a higher carbon steel is used.

G. SHERMAN.—Is there danger of grinding through the hardened shell to the soft steel inside or is the effect of hardening unusually deep?

C. W. NICOLSON (author's reply).—The depth of hardening of this bit is perhaps a little unusual, extending for ⅜ in. When the full life of the bit is obtained without regrinding, as occurs with about half of the bits, the decrease in length of the bit, due to the wear on the cutting edges, is not sufficient to go below this hardened zone. When, however, it is necessary to resharpen, it is true that the grinding goes through the hardened shell and it is therefore necessary to reharden. All bits to be sharpened are annealed and, after grinding, rehardened. The grinding of annealed rather than hard bits is simpler, faster and involves less wear of the grinding wheels. The grinding is done at the rate of about 60 bits per man per hour.

B. F. TILLSON,* Upper Montclair, N. J.—Have you tried the vacuum hood with much success or practicability?

C. W. NICOLSON (author's reply).—No, we never have. We have depended on wet drilling entirely. Samples taken alongside the drill do not show excessive dust. There may be some for the first foot or so of the hole but it seems to be rapidly dissipated. We have not tried some of the refinements because the dust counts with our present methods seem to be satisfactory.

T. T. READ,† New York, N. Y.—I should like to raise the question as to the relative responsibility of operating men and machinery manufacturers in the problem of arriving at the best and cheapest way of making holes in rocks. The makers of machinery have done splendid work in improving rock drills for that purpose. I wonder if operating men have not leaned on them a little too heavily.

Assume a basic setup of competitive machine manufacture. The better machine a firm makes, the easier it is to sell. One might say that all the operating man has to do is to let the manufacturers compete in a free economy. But under free competition in selling improved types of machines we might rely on the makers to continue to improve them and bring out the very best types.

The impression I received from Mr. Nicolson's presentation was that an important development, the new shape for a detachable bit, came from the operators, not from the makers of drills. Did the makers do anything more than just help? They did not originate the suggestion to improve the shape of the bit, did they?

C. W. NICOLSON (author's reply).—It has been my experience that, although manufacturers do a great deal of experimental work of their own initiative, cooperation between operators and manufacturers or encouragement of experimental work by both is necessary for the development of special designs that will solve the problems peculiar to different districts. This is, I think, particularly true in the matter of special designs of detachable bits, as the manufacturer does not have available the particular kind of rock for which the bit is intended and very few operators have the necessary facilities for making detachable bits, even for experimental purposes. It is only recently, I believe, that mining men have realized that, for maximum efficiency in drilling, each district should have its own design of bit and that the standard four-point cross bit, with 5° and 14° tapers, is, at best, only a compromise. This "Picher" bit was, I think, the first bit to be especially designed for the work it had to do but neither the operator nor the manufacturer can take sole credit for the design. We suggested some changes, the manufacturers suggested some, and so the design was evolved.

---

* Consulting Engineer.

† Professor of Mining Engineering, School of Mines, Columbia University.

T. T. Read.—The further point I am trying to make is that while the work done in the Tri-State district was excellent as a study of the problem under the conditions there, it does not necessarily follow that similar results would be obtained under different ones. In the Tri-State district most of the holes do not make a high angle with the horizontal, and if vertical holes were being drilled the attempt to avoid wear on bits by producing relatively large chips might be defeated by the greater difficulty of getting large chips out of a vertical hole as compared with a nearly horizontal hole. And who can say offhand whether the same results would be obtained from such a modification of practice while drilling in granite by the method used in drilling in chertified limestone? It does not seem reasonable to expect manufacturers of drilling equipment to study such problems, because they do not have the actual conditions available to them for laboratory study. Not every operator could or should make such an elaborate study of his own conditions as Dr. Suzuki* has done, but a group of operators might make a joint investigation; one that no single operator could afford to make because the probable benefit to him alone would not exceed the probable cost. But if the cost of the investigation is divided among all the people that benefit by it it would be profitable for all of them. I am pleading not only for more studies under actual field conditions but also for a method of financing them so that they may become economically profitable.

K. V. Lindell,† Copper Cliff, Ont.—I would suggest that there be some standardization in submitting data on such tests. We find it very important to know how fast a bit drills each minute—that is, if you record a run of 2 min. you may find quite a decrease in drilling speed in the next 2 min. The total efficiency of any bit must consider the labor efficiency as determined by drilling speed to complete any cost analysis and may have a bearing in maintaining a desired working cycle. So I would make a plea that there should be some basis of uniformity in the presentation of data on both bit and drill tests.

C. W. Nicolson (author's reply).—There is no question, of course, of the desirability of incorporating accurate data in these technical papers if such data can be obtained, and if they are of any value to the readers of the papers. In reports covering rock drilling such data could be obtained for homogeneous rocks but in rocks such as the brecciated flints of the Tri-State district, which vary greatly in hardness or ease of penetration, the results obtained in attempting to determine the speed of penetration vary between such wide limits that only averages for several hundred feet of hole seem to have any meaning. We once placed an observer at a drill for two weeks and recorded the speed of penetration for every change of steel, but the attempt to plot these readings produced extremely irregular curves. For instance, in one hole a sharp bit penetrated the first 24 in. in 1 min. whereas another sharp bit required 15 min. for the next 24 in. In fairly uniform flint, a sharp detachable bit has an average speed of penetration of about 11 in. per minute for the first 24 in., after which the speed drops to about 9 in. per minute and remains nearly constant for the remainder of the life of the bit. The slightly slower drilling speed of the dull bit is more than offset by other advantages obtained from the use of the detachable bit, and any attempt to use sharp detachable bits with each change of steel increases, rather than decreases, the total cost of drilling.

I believe that any attempt to present these results with figures other than averages would be confusing and perhaps misleading.

---

[*] T. Suzuki: An Experimental Study of the Rock Drill. See page 206, this volume.

[†] International Nickel Company of Canada, Ltd.

Ymir Yankee Girl Gold Mines Ltd.

BY LAWRENCE B. WRIGHT,* MEMBER A.I.M.E. AND LESTER G. MORRELL†

(Vancouver Meeting, September, 1937; New York Meeting, February, 1938)

ON the steep slopes of the Kootenays, 15 to 20 miles south of Nelson, B. C., are a number of ore occurrences (Fig. 1). One that has many features of mining and geological interest is the Ymir Yankee Girl Gold Mines Limited.

Discovery of this property was made in 1899 at an outcrop on the hillside, some 2500 ft. above the village of Ymir. This proved to be a branch vein 20 ft. long leading to two main veins, the Yankee Girl and the Spur (Fig. 5).

The ore pattern as now delimited shows a remarkable increase in ore area with depth below the outcrop. This recent development is of interest particularly in view of the fact that the property was nearly condemned, professionally. Another interesting feature, which has a bearing upon its history, is the persistent excess of tonnage produced from period to period over visible reserves. This is not ascribed to lagging mine development, but to detailed vein characteristics, which we will endeavor to make clear by giving a description of the veins and their surroundings.

HISTORY AND PRODUCTION

Little interest was evidenced in this district until the building of the railway in 1893 provided easy access and the discovery of the Ymir mine‡ in 1896 attracted attention. As a result, the Ymir Yankee Girl group of claims was staked in October 1899.

In 1907, an American syndicate took over the property and subsequently extended the 50-ft. drift and opened the present 400 and 540-ft. levels. In 1908, the Syndicate, without recording any shipments in the meantime, dropped its lease, which was taken up by H. L. Rogers, who bonded the Yankee Girl group. Rogers shipped 2622 tons of ore, mainly from the uppermost stope in the mine, which is reported to have yielded

---

Manuscript received at the office of the Institute Sept. 10, 1937; revised Dec. 30, 1937. Issued as T.P. 937 in MINING TECHNOLOGY, July 1938.

* Mining Geologist and Consulting Engineer, Toronto, Ont.

† Superintendent, Ymir Yankee Girl Gold Mines Ltd., Ymir, B. C.

‡ Not to be confused with present Ymir Yankee Girl mine.

1.18 oz. of gold per ton. After further shipments, amounting to 4738 tons, ownership of the property was transferred to the Hobson Silver Lead Co. Ltd.

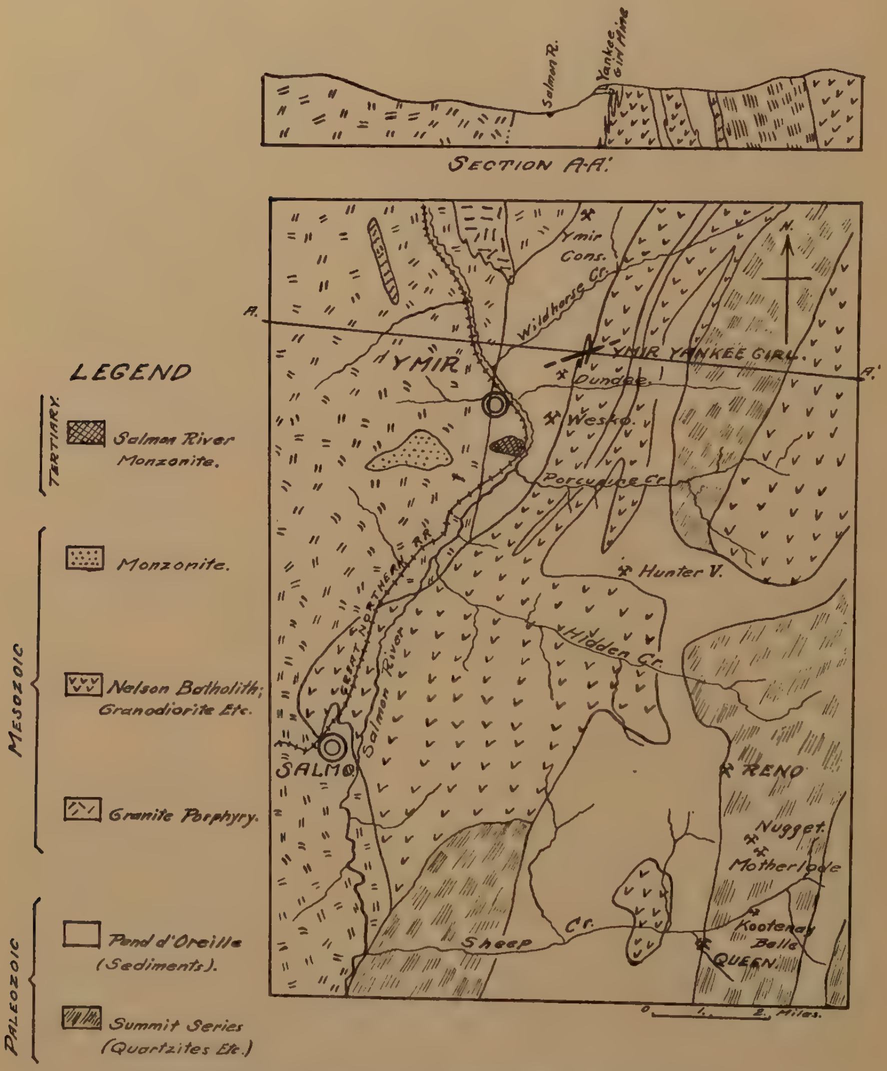


FIG. 1.—GEOLOGY OF YMIR-SALMO DISTRICT.
Adapted from maps by C. W. Drysdale and J. F. Walker.

During the years 1912 to 1919, the mine plant was enlarged and a 6000-ft. aerial tram was erected. Several thousand feet of development was done including the 4000 ft. of drifting on the 1235-ft. adit level. During the period named, 16,500 tons of ore was shipped, averaging

0.82 oz. gold per ton, principally from the McDowell stope on the 1235-ft. level.

In 1919, a large operating company became interested and optioned the property for examination. The option was dropped on the grounds that the ore was of insufficiently high grade. It is interesting to note that the final report of that company shows: probable ore, 25,590 tons; possible ore, 63,700 tons. Nearly twice the combined total has been shipped or milled since.

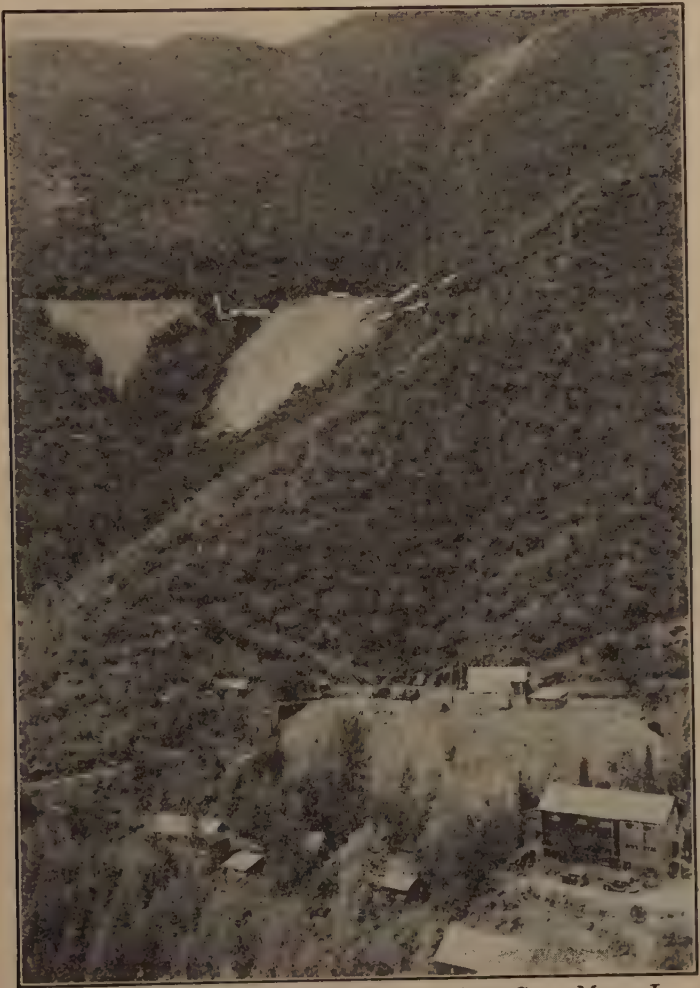


FIG. 2.—MINE AND CAMPS, YMIR YANKEE GIRL GOLD MINES LTD.

In 1926, F. R. Weekes, E. P. Crawford, W. T. McDowell and others formed the Yankee Girl Ltd. to operate the property under lease and bond. Their efforts produced 28,358 tons averaging 0.57 oz. of gold per ton. They shipped 1251 tons of ore that yielded 0.43 oz. of gold per ton.

Weekes and Crawford again leased in 1932. With the help of the premium on gold, and in spite of low lead and zinc prices, their venture was sufficiently profitable to permit opening much new ore. Under their partnership lease, 32,433 tons averaging 0.57 oz. gold per ton were shipped.

The present company was formed in 1934. Shipments prior to August 1934 were 87,529 tons averaging 0.68 oz. gold per ton. Between August 1934 and July 1, 1937, 94,106 tons have been milled, averaging 0.34 oz. gold per ton. In all, 181,635 tons have been produced, averaging 0.50 oz. gold per ton.

GENERAL GEOLOGY

Drysdale,[1] Walker[2] and Cockfield[3] have mapped and described the geology of the area in which the Ymir and Sheep Creek ore deposits occur. Considerable of the following material bearing on stratigraphy has been taken from their publications.

TABLE OF FORMATIONS

TERTIARY	Pleistocene	Glacial till, silt, sand and gravel.
	Eocene-?	Salmon River monzonite.
MESOZOIC	Jurassic	Nelson batholith, tongues, stocks and dikes.
	Triassic	Beaver Mountain-Rossland group. Mostly volcanics.
PALEOZOIC-?	Cambrian-?	Pend Oreille series. Sediments.
PRE-CAMBRIAN	Windermere	Summit series. Mostly sediments.

The oldest rocks of the vicinity are the pre-Cambrian Summit series, consisting of argillites, limestones, grits, conglomerates and several thick quartzite members. Among these the Reno Formation and the Quartzite range are of economic importance because their brittle nature has favored the occurrence of fissures in which ore-bearing veins of the Sheep Creek district are found.

The Pend Oreille series conformably overlies the Summit series. It is either late pre-Cambrian or Cambrian in age and is composed of fine-grained sediments, slates, phyllites, limestones and quartzites, all characterized by carbonaceous material. In places where shearing has been intense, graphitic schists are formed. This series being less competent during mountain building, is consequently highly folded and contorted.

The Beaver Mountain-Rossland group is composed largely of volcanic rocks: greenstone, ash rocks, volcanic breccias and conglomerates. The last activity during this period gave rise to a series of granite porphyry dikes and masses, which occur, in part, north of Ymir. A distinct north-south fracture system resulted, to be later confused by the intrusion of the Nelson batholith and later intrusives. This system, under favorable local conditions, serves as secondarily important loci of mineralization. This series overlies the Pend Oreille beds, probably unconformably.

---

[1] C. W. Drysdale: Ymir Mining Camp, B. C. Canadian Geol. Survey *Memoir* 94 (1917).

[2] J. F. Walker: Geology and Mineral Deposits of the Salmo Map-Area, B. C. Canadian Geol. Survey *Memoir* 172 (1934).

[3] W. E. Cockfield: Lode Gold Deposits of Ymir-Nelson Area, B. C. Canadian Geol. Survey *Memoir* 191 (1936).

Rocks of the Nelson batholith vary in composition and texture. They consist of granite, porphyritic granite, aplitic granite, granodiorite, quartz diorite, syenite and monzonite. All of the older sedimentary and volcanic series have been intruded by one or more phases of the Nelson batholith. Stocks and tongues outcrop over a large portion of the area, occurring as long parallel intrusions of widely variable dimensions. Remnants of the Pend Oreille and Summit series, into which the granitic mass has been intruded and which formed the roof of the batholith, now outcrop as a series of elongated parallel bands.

Lamprophyre dikes are the youngest rocks in the district and cut all other rocks as well as mineral deposits.

Recent deposits consisting of unconsolidated glacial till, silt, sand and gravel, cover the bottom of Salmon River Valley.

SEQUENCE OF PHENOMENA LEADING TO ORE DEPOSITION

1. *Sediments* of the Pend Oreille series were laid conformably upon the earlier Summit series. The presence of organic matter is evidenced by the resulting carbonaceous nature of these beds.

2. *Volcanic debris and lavas* flowed over the sediments to a thickness exceeding 2000 ft. The pressure exerted by this material probably served to aid consolidation of underlying sediments.

3. *Granite intrusion* on a batholithic scale followed. Where bedding in the invaded sediments was pronounced, true lit-par-lit injections took place. Zones of intense pressure resulted in schisting. Contortions in schist laminae give evidence of the plastic condition of the rocks. Stresses were apparently relieved by flowage rather than faulting, at least until near the termination of igneous activity (Fig. 3).

4. *A period of fissuring and faulting* followed the batholithic intrusion as a result of contraction and differential adjustments of the affected areas. It is believed that many of the present vein fissures were formed during this period.

5. *First period of vein mineralization:* The ascent of mineral-bearing solutions commenced soon after the rocks of the district were relatively cool and continued throughout a considerable period, the chemical content gradually changing. The earliest solutions deposited quantities of quartz and pyrite.

6. *Movement along the three major fault planes* has not been great in the Yankee Girl mine, but activity has been more or less continuous to recent times. The veins within the fault planes include fragments of

wall-rock schist and granite. As mineralizing solutions changed in composition, their contents were deposited in new fissures within the older veins, thus forming compound veins. In addition, lamphrophyre dikes, which in places traverse along and occasionally cut the veins, are themselves cut by gouge-filled fissures, which are characteristically associated with the vein planes, thus indicating that movement along the vein faults has been continuous.

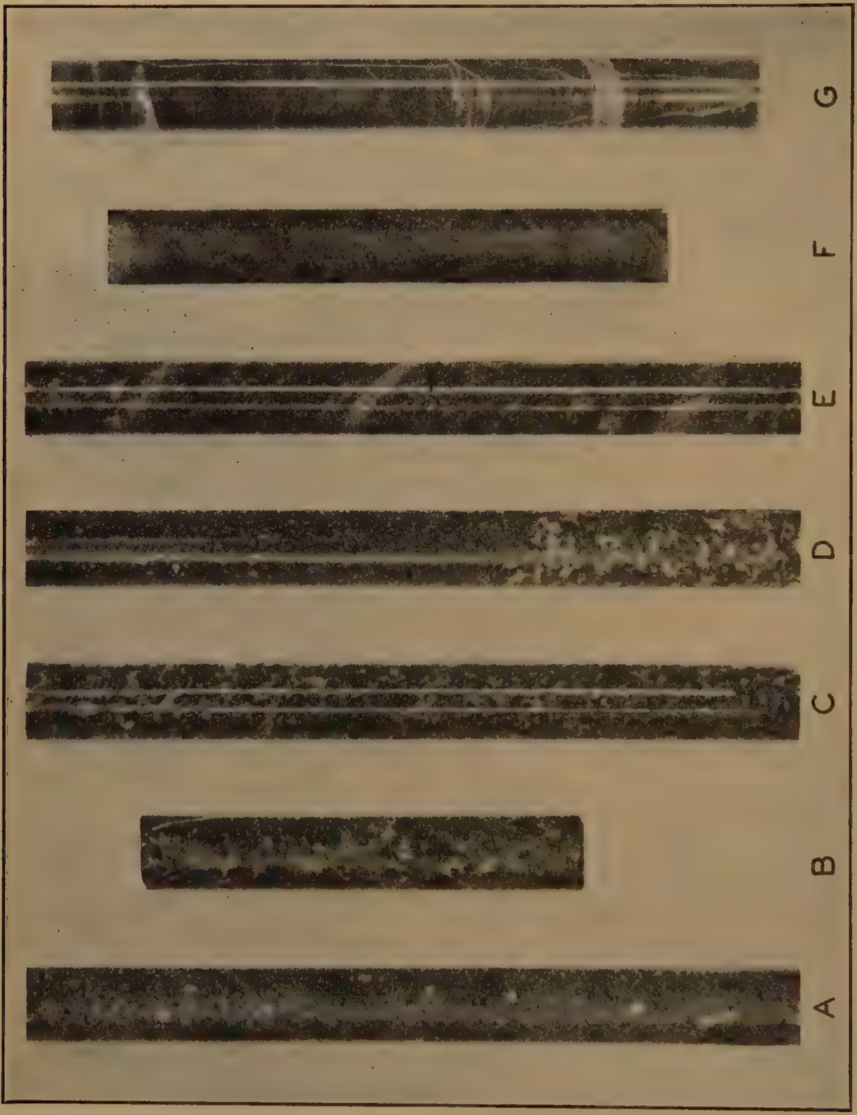


FIG. 3.—DIAMOND-DRILL CORES, SHOWING ROCK TYPES AT YMIR YANKEE GIRL MINES.

A. Diorite phase of Nelson granite. *B*. Extremely coarse-grained phase of Nelson granite. *C*. Normal granite and granodiorite. Note slight gneissic structure. *D*. Contact of two granitic phases. *E*. Specimen showing lit-par-lit composition. Note three "beds of granite." *F*. Schisted sediments. Intensity of flowage is shown in this specimen. *G*. Typical specimen of sediments.

7. *The second period of mineralization* could perhaps be more accurately referred to as the "commercial portion" of one protracted period. The later solutions deposited quartz, pyrite and subordinate quantities of gold, silver, lead and zinc, all of which combined constitute commercial ore. Despite the difficulty of ingress, the second-period minerals were controlled in distribution by permeable areas and fissures remaining available in the existing veins. Continued movement and hydrothermal effects along the vein walls resulted in "high-grade" bands that occur as component parts of the vein structure. They lie on either wall or occupy an intermediate position within the first-period vein matter. A few spur fissures in the walls are found to contain profitable material of the later mineralization. Fig. 4 is designed to show graphically the sequence of change in the composition of vein filling throughout the entire period of mineralization.

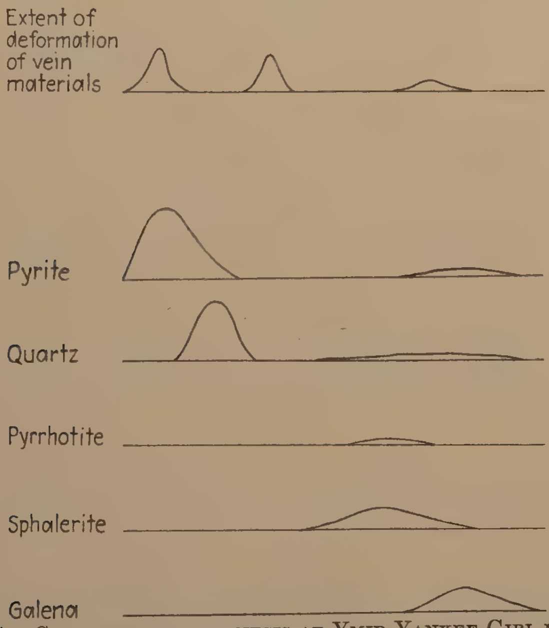


FIG. 4.—SUGGESTED PARAGENESIS AT YMIR YANKEE GIRL MINES. After D. M. McKinnon.

8. *Numerous aplite and lamprophyre dikes* were intruded into the rocks of the region after approximately present conditions obtained. The latter are found cutting across veins and country rock fortuitously. Apparently they have not affected the ore.

9. *Erosion* has made possible the discovery and ultimate development of the Yankee Girl mine. In this particular instance, nature was not particularly generous, for the only known outcrop is less than 20 ft. long. Normally, intense *oxidation* of vein matter has not appeared much below

the stopes on the 540-ft. level but extends along water courses throughout the mine to a depth of 1500 feet.

MINE GEOLOGY

Structure

Yankee Girl ore deposits are of the fissure-vein type, having been localized in faults cutting the granite-schist contact zone. The Yankee Girl and Spur veins, both of which contain commercial ore, are found in fault planes that cut obliquely across the planes of schistosity. The Lakeview, a noncommercial vein, occurs within the Lakeview fault fissure, which parallels the formation in a long narrow Pend Oreille remnant enclosed between thick granite tongues (Fig. 5).

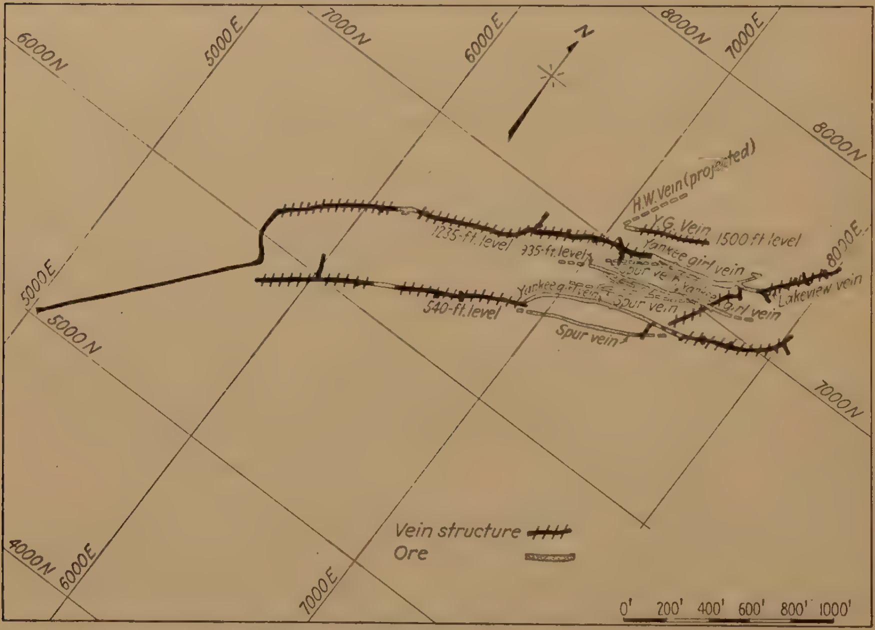


FIG. 5.—YMIR YANKEE GIRL MINE.
Composite plan, 540-ft., 935-ft., 1235 and 1500-ft. levels.

The intruded granite members have aligned themselves along the planes of schistosity, forming a complex alternation. Several bands of both rock types have often been found in 12-in. pieces of diamond-drill core.

Development in the mine has shown that in general the invasion zone is irregular. So far as development has extended on the lower levels, the inference might be drawn that the granite mass dips steeply to the east, although Fig. 6 indicates a westward dip. Bodies of granite that have penetrated schist assume the attitude of the schist planes, which dip, in

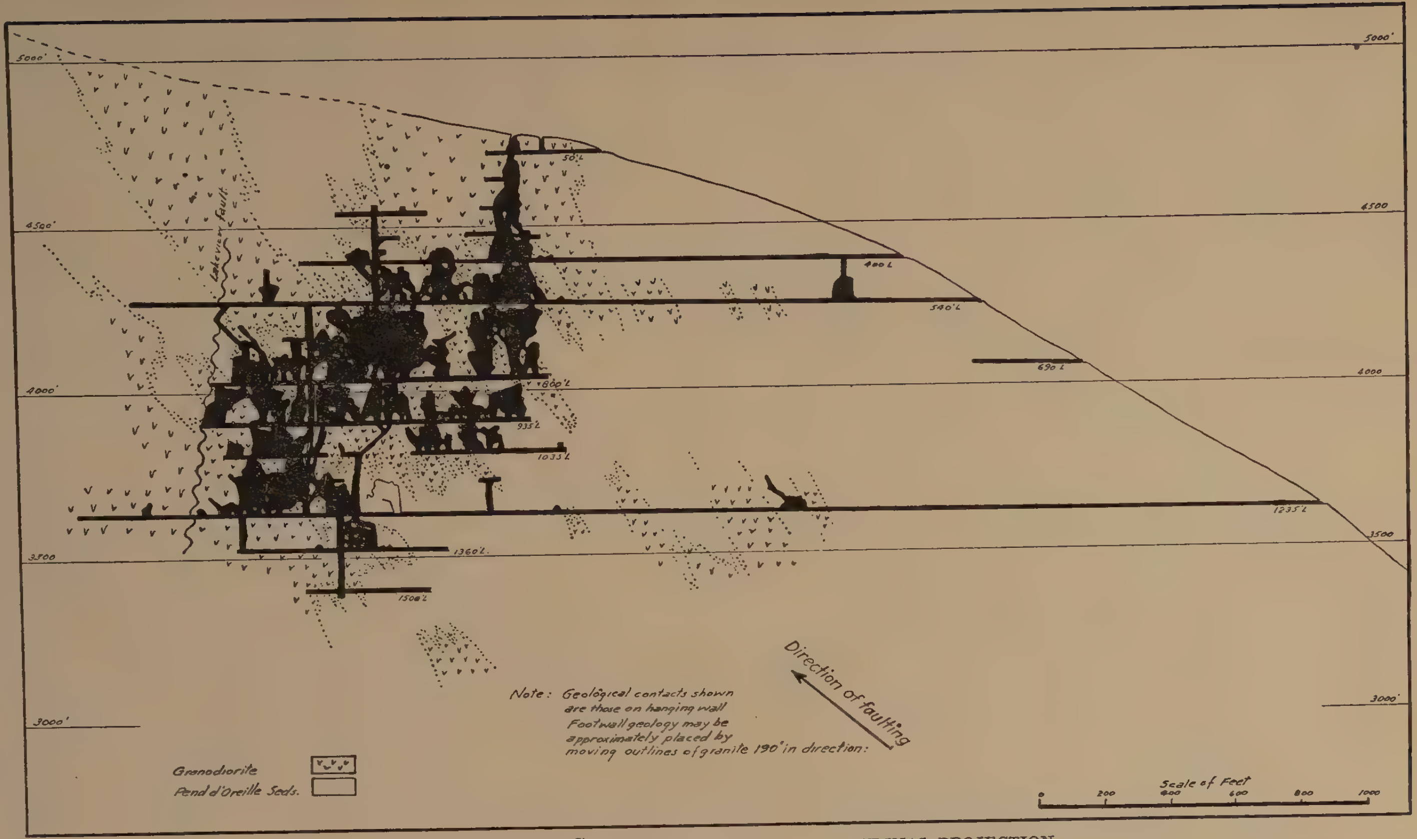


FIG. 6.—YANKEE GIRL VEIN, VERTICAL LONGITUDINAL PROJECTION.

the vicinity of the Yankee Girl mine, 60° to 70° toward the northwest along a strike of N. 35° E.

The Yankee Girl-Spur vein fault planes are roughly parallel and strike N. 65° E. with a dip to the northwest. The trace of schisting and granite-tongue intersections with the veins is to the west, dip 50°. Solutions may have been controlled by these prominent vein-wall features.

The Yankee Girl-Spur vein system may be regarded as a single unit. The combined components are terminated at their intersections with the Lakeview fault at the east end of the known commercial ore bodies. There is some evidence in support of the view that the Lakeview vein does not actually fault the Yankee Girl, the ore being limited structurally instead.

The direction of movement along the Yankee Girl-Spur fault zone is readily determined by striae on the walls. The hanging-wall block has moved (with respect to the footwall) down and to the west at a vein plane angle of 50° through about 190 ft. Such a movement is indicated by the intrusion pattern shown in Fig. 1. Post-mineral movement, that is to say, post-all-mineral, has occurred, but its effects have been slight. Cross-faulting of the Wesko vein system one mile south is attributed to the Tertiary monzonite mass west of the mine, shown in Fig. 1. This mass is evidently bounded on the north by the earlier "vein system" fractures. Its effects are not extended across the previously uplifted block to the Ymir Yankee Girl vein.

ORE BODIES

In general, the present stoped areas of the mine are indicative of the position of commercial ore throughout the mine workings. In addition, large blocks of virgin ground, like that west of the present extent of development, east of the outcrop and above the 400-ft. level and deeper horizons, are all considered favorable areas. (See Fig. 6.)

Invariably veins are found filling fissures of greater or less magnitude that occur as faults or joints. As a rule, ore bodies are not liable to be found along prominent or open breaks. This is accounted for by relative lack of impounding of solutions locally and excessive permeability. Ore is far more often deposited in adequately permeable paths such as breccia-filled fissures, or in tight joint planes where circulation was a slow process and ample time for penetration and precipitation was allowed. For example, the low-grade Lakeview, especially north of the Yankee Girl intersection, gives evidence of having been a more open fissure than the Yankee Girl or Spur.

Control (and Localization)

Ore bodies occur in the Yankee Girl mine within the lit-par-lit zone bordering the west side of a wide granite tongue. The Lakeview fault plane cuts through massive granite, and in extending southwest passes through the lit-par-lit zone *along* the planes of schistosity. The Yankee Girl and Spur planes, on the other hand, cut the schist at an acute angle. It is concluded therefore that fissures favorable for ore deposition are those that cross the schist planes and cut concordant granitic dikes. A detailed study of the variations in strike along the Yankee Girl fault plane on various levels shows that where dikes are involved the plane is warped usually into the hanging wall. The best ore occurs in these gentle warpings. Structurally, they rake westward along the plane and account for the several local westward-raking segments of the orebody.

Regarded as a whole, ore deposition occurs in an eastward-raking zone, although work in depth may show the productive area to have a major westward rake controlled by intersection of dikes and vein planes. Both possibilities will need to be investigated and could both result in furnishing loci for ore.

It is believed by the writers that disseminated carbon present in the sheared faces of carbonaceous sediments, and dragged into the dike area, acted as the chief precipitant effecting deposition.

Early mineral-bearing solutions, characteristically depositing large quantities of barren pyrite and quartz, are found to have penetrated most of the area covered by the Yankee Girl and Spur as well as the Lakeview veins. Continued or intermittent movement along the shear zones maintained a permeable condition throughout the entire period or periods of mineralization. As mineralizing agencies transported the more valuable minerals, these were deposited in the hospitable areas, as evidenced by the high-grade streak so common in commercial stopes, occurring along either side or within the older filling.

The Spur vein is a prominent footwall component of the Yankee Girl shear zone. It occurs in a flat concave fissure beneath the Yankee Girl vein. Its rim is more or less elliptical in outline with a major axis plunging at an angle of 70° to the east for a length of 800 to 900 ft. The minor axis has not been definitely determined, but measured on the 800-ft.level, it appears to be approximately 850 ft. long.

Countless numbers of minor breaks in the rock between the Yankee Girl and Spur veins have been filled with vein matter. Most of these display little or no fault movement and few are of commercial interest; the outstanding exceptions to the latter being the "intermediate veins" on the 800-ft., 935-ft. and 1035-ft. levels; also the "hanging-wall veins" on the 1235 and 1360-ft. levels.

ORE-SHOOT PATTERN

It has been pointed out above that the lit-par-lit zone controls the locus of ore. It has also been shown that the favorable area of alternate sediments and dikes trends eastward. Logically, then, since the control dips in that direction, the general trend of ore occurrences as a whole may likewise be expected to plunge to the east, while some ore shoots within the zone boundaries rake to the west.

It would appear from this pattern, when also considering the broad westward-plunging flutings in the hanging wall, that solutions ascending along these channelways encountered the general eastward-raking favorable area where precipitation was stimulated.

Origin of Solutions

We recognize three possible sources of mineralizing solutions:

1. Nelson granite in its several phases.
2. Post-batholithic intrusives (monzonite stocks and aplitic differentiates).
3. An unexposed later magmatic source.

That the ore deposits of the region are broadly related to the Nelson batholith is generally accepted. Included within a probable long period of mineralization are ores in other mines that were deposited commencing at the time of the first quartz-pyrite mineralization at Yankee Girl. There must also be considered those later deposits that represent the final phase of mineralization of which the ores are composed largely of silver, lead and zinc.

The fact that mineralization followed the faulting of Nelson batholith apophyses fixes this event at a time when these dikes were in place, relatively cool and faulted. The intrusive next following the batholith and attendant differentiates is the monzonite mass south of Ymir. If, as the pattern of fracturing suggests, this mass created the vein openings, the solutions then followed, but are not necessarily directly related.

The Tertiary Salmon River monzonite appears to be post-mineral, if we accept the probability that its invasion caused the cross faulting of the Wesko ores.

As to the third possibility, the effects of unexposed (Cretaceous?) volcanics may have some influence. The periods involved are so great that we can only give the apparent sequence of events and conclude broadly that:

a. The ore solutions were late differentiates of the Nelson magmas, or

b. They were introduced considerably after the end of Nelson batholithic activity and prior to the Salmon River (Tertiary) monzonite.

Vein Mineralogy

The over-all average *vein width* approximates 4½ ft. Frequent variations from this width range from 30 ft. in one place to a few inches of high grade in narrow stopes and to stringers of near microscopic size penetrating the wall rocks.

The walls are usually clean, permitting extraction of the entire vein without including wall material. Within the vein itself, cleavage planes parallel to the walls often cause concern by being mistaken for true walls. In a few cases mineralization by replacement permeates one wall or another, more frequently the footwall. In these places assays control mining limits.

True quartz veins are rare. The hanging-wall vein on the 1360-ft. level is the best example of this type. The quartz is pale gray to white in color, friable and brittle. Here the pyrite to galena-sphalerite ratio is high. The sulphides are of rather coarse grain and generally pale bronze in shade. As a rule vein matter of this type seldom exceeds a normal grade of 0.60 oz. gold per ton.

The Yankee Girl and Spur veins are mainly sulphide-bearing silicified schist veins. Fragments and laminae of schist (and granite) are included in the vein matrix. Such inclusions are "soaked" with mineralizing agents. Films of graphite are commonly associated with this type of vein, especially along the faulted walls.

Here all forms of commercial veins characteristically show a high-grade streak. The lead-zinc later-period mineralization thus represented occupies a compact band meandering from wall to wall. These streaks are literally black with sulphides of iron, lead and zinc, and hence readily discerned. Some areas, particularly in the Spur vein, are mineralized exclusively with high-grade material of this type. Ore bodies in these situations are usually narrow, a foot or less to 2 ft. wide, and the ore produced from them usually assays 1 to 4 oz. per ton in gold.

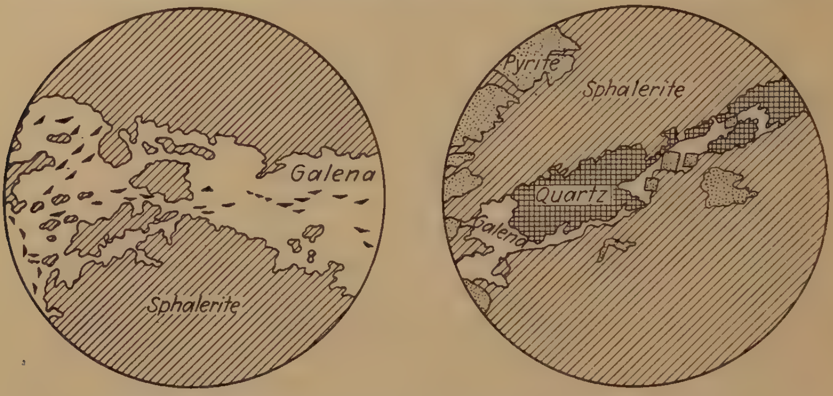
A post-mineral clay and breccia gouge varying in width from a fraction of an inch to two or more feet is always, in both the Spur and Yankee Girl veins, associated with the vein structure. Particularly common in the Spur vein is a lamprophyre dike 2 to 18 in. wide. Usually it lies adjacent to either wall, so that by careful mining it can be left in place.

An average analysis of mill heads, which shows the average value, is: Au, 0.34 oz. per ton; Ag, 1.4 oz. per ton; Pb, 1.5 per cent; Zn, 3.13; Fe, 9.4; SiO_2, 52.0 per cent.

The metallic minerals are: native gold, galena, sphalerite and pyrite. In addition to these are the rarely found minerals: pyrrhotite, chalcopyrite and oxidation products of lead and zinc. Quartz and calcite are the chief gangue constituents, exclusive of the small quantities of other accessory nonmetallics contributed by wall-rock fragments.

Native Gold is reported to have been abundant in the surface showing and in the Hobson stope but during recent years a specimen showing visible gold has not been seen. The silver-gold ratio in the average run of mine assays approximates 3:1.

Galena occurs in a range of grain sizes from fine-grain "steel" variety to a coarse structure in which cleavage faces measure ½ in. across. The smaller crystals indicate a higher gold content than do the large ones. A fine-grain complex intergrowth of galena, sphalerite and pyrite in a matrix of quartz is the combination required for maximum gold values (Fig. 7).

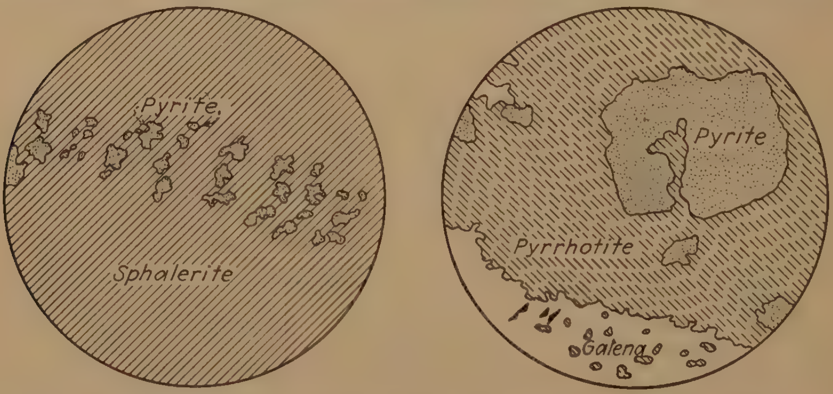


a *b*

FIG. 7.—SKETCHES OF POLISHED SURFACES OF YMIR YANKEE GIRL ORE. × 50. After D. M. McKinnon.

a. Specimen Y-1. Galena veinlet in sphalerite. Note mutual boundary relationship between the two minerals.

b. Specimen Y-1. A second generation of quartz and pyrite is suggested by this veinlet of galena, quartz and pyrite cutting sphalerite. Note euhedral development of pyrite in veinlet.



a *b*

FIG. 8.—SKETCHES OF POLISHED SURFACES OF YMIR YANKEE GIRL AND WESKO ORES. × 50.

a. Specimen Y-2. Appearance of one peculiar pyrite "veinlet" cutting sphalerite. Embayed nature of pyrite borders suggests that it is the older mineral.

b. Specimen W-1. This figure suggests that pyrrhotite is later than early pyrite. Note mutual boundary patterns and emulsion structure between galena and pyrrhotite.

Sphalerite, several varieties as well as a limited range of grain size, is observed in this mineral. As a rule, specimens of apparently pure sphalerite, regardless of color or grain size, are wanting in gold content, but occasionally carry silver in small amounts. Under the microscope sphalerite is seen to absorb pyrite, thereby suggesting an origin for the isomorphous iron present in the dark brown and black varieties. Mutual contact relations between grains of galena and sphalerite are common. (Fig. 8.)

Pyrite is a particularly abundant mineral, occurring both in coarsely crystalline masses measuring up to several feet across, and as extremely small isolated grains. Pyrite is a host for gold when the grain size is small and when quartz is present. Highest values are found where pyrite is intimately associated with other sulphides. Under the microscope, pyrite develops crystal form in galena and quartz but takes allotriomorphic habit in sphalerite. Textural relations confirm the theory of two generations (Fig. 8).

Chalcopyrite is a comparatively rare mineral in Yankee Girl ores. It is seen occasionally as tiny masses or veinlets in quartz or associated with pyrite.

Pyrrhotite occurs in specimens of ore found in the lowest levels of the mine. It is not particularly common. Pyrrhotite is a chief constituent of the Sheep Creek ore bodies and has been found in Wesko ores (Fig. 8).

Quartz is the chief component mineral of the ore. It is white to bluish gray in color. The latter type is characteristically associated with ore of better grade. The microscope shows it to occur as irregular fragments with rounded or scalloped borders in galena and sphalerite. Since quartz has conchoidal fracture, this does not imply contemporaneity with its surrounding minerals. The second generation of quartz is suggested by its association with galena and pyrite in veinlets enclosed by sphalerite (Fig. 7).

The sulphide associates of gold have been indicated by assays of selected specimens of the ores:

	Oz. Au per Ton
Pure pyrite of first generation with quartz	0.08
Pure sphalerite	Trace
Pure galena	0.64
Pyrite, galena, sphalerite	1.80

Examination of a small suite of ore specimens in polished surface has established a paragenesis which confirms previously stated views.

DISTRIBUTION OF GOLD IN THE ORE ZONE

Occasional high-grade samples (5 oz. gold per ton or better) are encountered uniformly distributed throughout the mine workings. As can be seen from the longitudinal section (Fig. 6) mine stope openings have assumed the shape of a rough ellipse, the upper and lower ends of which have not been fully explored laterally. Throughout the vertical extension of the known ore area a tendency toward higher grade is noticed in the upper portion than in the lower. The middle section is of slightly lower grade than either end but has produced the bulk of the ore by reason of a greater tonnage. An interesting feature is that ore is seldom found to parallel itself; i.e., where the Yankee Girl vein carries good values, veins in the hanging wall or footwall are comparatively barren, and vice versa.

Mining and Development

Mining Methods.—The vein matter being comparatively soft, is not difficult to mine. It is often associated with a gouge seam that provides a cleavage plane to which the ore can be broken. The walls as a rule are cleanly marked and the vein material readily separates. Where the necessity for mining a horse of waste arises, it is broken large to facilitate sorting at the chutes.

The erratic distribution of gold within the veins requires that a most careful watch be kept on stoping. Ore areas of the vein are irregular in outline and size. For this reason a selective method of mining has been adopted. The outlines of stopes are uneven and pillars are numerous. Bodies large enough to extract vary in height from a few feet to 700 ft., and in length to 500 ft. The widths range from a foot to a maximum of 15 feet.

Underground Exploration.—Prospecting and exploration are carried on by a combination of drifting, crosscutting, raising, winzing and diamond-drilling. The usual procedure is to drift along the main or Yankee Girl fissure. From the drift, diamond-drill holes are drilled 50 to 200 ft. into the wall. Special attention is given to the footwall (southwest) side, which is known to contain the Spur vein. A raise or two explores the area above the drift and when connected to a level above aids ventilation. Sublevels are often driven from raises to explore areas known to be favorable for ore deposition. Crosscuts are seldom driven unless they are intended to open an ore body that has been found by diamond drilling.

Sampling and Estimation of Tonnage and Value.—Each new development face is chip-sampled to indicate current values, assays being made for gold only.

Stope faces are sampled over the full vein width every round.

Cars of ore or doubtful material are sampled by a grab from each car. This composite sample is quartered and assayed daily.

Diamond-drill core is logged, abstracted every 5 ft. or less, and sections of core are split for assay and permanent record. Sludges are discarded when the core indicates that waste has been drilled. They are assayed if the corresponding footages of core show mineralization.

All stope and raise samples are plotted on 20-ft. scale vein-plane sections. The sampling of drifts and crosscuts are plotted on 20-ft. scale sections and on 40-ft. scale plans.

Monthly production from the various working places is estimated by applying a factor, ranging from 0.8 to 0.9, to the number of cars drawn from a working place, thereby estimating tonnage; and geometrically averaging the daily car samples for grade of ore. An estimate of the tonnage and grade of broken ore is calculated by monthly stope measurements and averaging the chip samples that have been taken during the period. All estimates of tonnage are based on the following factors: One ton requires: ore in place, 11.0 cu. ft.; waste in place, 12.0 cu. ft.; broken ore, 18.0 cu. ft.; broken waste, 20.0 cu. ft. Assays are reported in ounces of gold per ton.

Development Details.—The property has been developed through three adits, the 400-ft., 540-ft. and 1235-ft. levels. At the present time the 1235-ft. level is the main working and tramming level and the mine plant is situated at its portal. Stoping levels include those at horizons 250 ft., 400 ft., 540 ft., 620 ft., 800 ft., 935 ft., 1035 ft., 1235 ft., 1360 ft. and 1500 ft. Practically all of the development work has been driven with the key objective of following the veins.

Drifts and crosscuts are 6 by 8 ft. in section and are driven at a slight (0.3 per cent) grade to facilitate drainage. In rare instances, where roof support is necessary, standard drift sets of round timber and lagging are used.

The standard raise is three-compartment consisting of two 5-ft. chute compartments on either side of a timber 6-ft. manway and slide compartment. It is driven the width of the vein or a minimum of 3 ft. This type of raise has the advantage of being readily changed to handle ore and waste, respectively, in the separate chute compartments—a distinct and useful advantage in this type of ore deposit. Occasionally small raises consisting of but one chute compartment and a manway are used for special purposes through ground that is unlikely to penetrate material requiring segregation.

Stoping Details.—The firm walls and vein matter are well suited to either shrinkage or cut and fill methods of mining, both of which are used. In a few instances both methods have been used in a single stope, the change being made midway between levels. It is necessary, in choosing which method shall be used, to allow for leaving waste; in pillars in shrinkage stopes or as fill in cut-and-fill stopes. Most of the mining in the past has been by shrinkage stoping.

The backs of drifts for shrinkage stopes are generally timbered with stulls placed at 5 ft. 6-in. centers and lagged with round poles. Chutes are on the footwall side and are spaced 11 to 22 ft. apart. Two manways are provided, one at each end of the stope. If the stope extends over

100 ft. high a timber slide is used in one of the manways. If stoping indicates that ore extends beyond the ends of the stope, box holes are put in the drift back and inclined to the stope or to each other, a chute is installed in each, and the length of the stope thus extended. Broken ore is maintained at a height of about 7 ft. below the back. Pillars of low-grade material are left where necessary to maintain the maximum grade of ore or to support large slabs of wall rock.

Cut-and-fill methods are adopted where the drift back shows a single narrow high-grade vein (2 ft. or less in width) or two or more such stringers separated by bands of waste. The drift is timbered, chutes are spaced 16 to 22 ft. apart and are built of framed 2-in. rough plank. The ore is blasted on plank floors and is sorted by the shovelers, who shovel directly into chutes. Ordinarily sufficient fill is obtained by regulating the width of the stope in relation to the width of ore. Where the vein itself does not contribute sufficient waste, it is slashed from the hanging wall.

Shrinkage stopes are converted to cut-and-fill by erecting a light sprag-and-plank floor at the height desired, timbering as for a drift back with chutes and manways, then drawing the lower shrinkage-stoped area empty. The cut-and-fill chutes are then drawn from the subfloor directly into the empty stope. Cut-and-fill stopes are changed to shrinkage by timbering from the fill floor with chutes aligned over the cut-and-fill planked chutes. Ore from the shrinkage stope is then drawn directly into cut-and-fill chutes.

Conversion from one method to another is, as can be readily seen, an expensive operation. It is avoided generally, but where the level interval is great, as between the 1235 and 1035-ft. levels or between the 800 and 540-ft. levels, it is sometimes advisable. (Fig. 9.)

Underground Transportation.—All loading and upper-level tramming is done by hand. Flat-bottomed 16 or 20-cu. ft. cars that dump either end or side are used. Ore and waste from the upper levels are carried to the 1235-ft. haulage level by a system of ore and waste passes, utilizing the large capacity of empty stopes and a few raises driven for the purpose.

Ore and waste from the 1360 and 1500-ft. levels is hoisted in a 1-ton self-dumping skip. The bin into which the skip empties is equipped with a gate that controls discharge into ore or waste compartments.

Two 1½-ton storage-battery locomotives are used in conjunction with cars of 1½ tons capacity to transport ore and waste to the surface.

Timber and supplies are hoisted through a vein-dip raise equipped with a wooden skip and hoist located on the 800-ft. level. Tugger hoists operate smaller skips between the 800 and 400-ft. levels.

Drainage.—The mine is not unusually wet, normal drainage amounting to about 100 gal. per min. During the spring break-up, the quantity is increased to three times the normal amount for a period of two months. Water flows to the surface in drainage ditches on the 540 and 1235-ft. levels.

Ventilation.—Ventilation in the mine is almost wholly natural, aided by compressed air exhausted from drills, pumps and hoists.

Aerial Tramway.—Ore brought to the surface is dumped into a bin of 100 tons capacity near the mine portal. This bin is constructed over the upper terminal of an aerial tramway which carries the ore, by gravity, to the mill. The length of the tram is 6000 ft. and the vertical drop between the upper terminal and the discharge bin is 1210 ft.

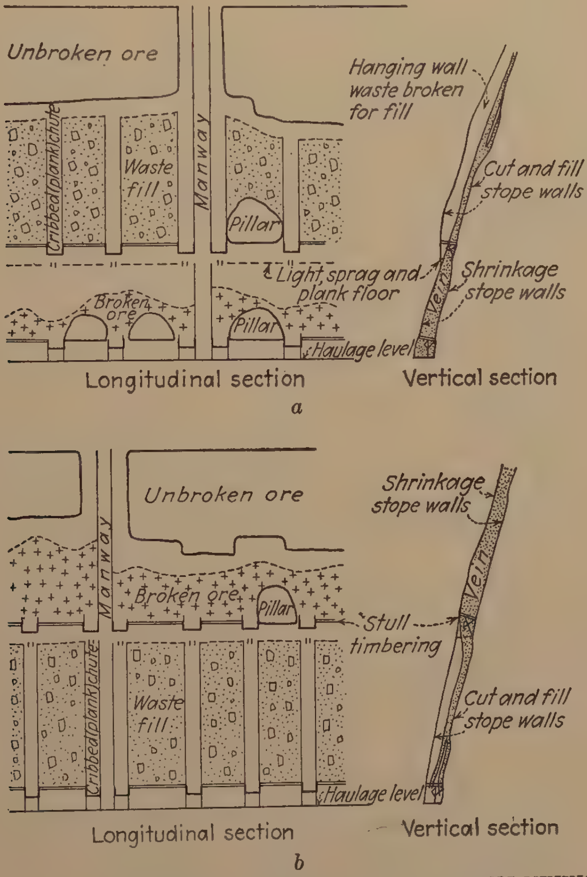


FIG. 9.—CONVERSION OF STOPING METHODS DURING MINING.
a. Conversion of shrinkage to cut-and-fill.
b. Conversion of cut-and-fill to shrinkage.

Seven towers are required to support the tram cables. The longest single span is approximately in the center of the distance and is 2000 ft. long. An unusual feature of the tram is that the unloading bin is at the base of a steep hill while the cable anchors and lower terminal are some 500 ft. farther out, across the Salmon River. This necessitates a hold-down tower on the hillside adjacent to the discharge point. The arrangement works very satisfactorily (Fig. 10).

TABLE 1.—*Mining Costs Ymir Yankee Girl Gold Mines Limited, Ymir, B. C.*

AVERAGES FOR THREE MONTHS, 1937

Cost	Labor	Supplies	Power	Total Cost	Tons Milled	Cost per Ton Milled	Supplementary Unit Costs
Development and preparation	$1740.39	$ 801.02		$2541.41	3357	0.758	9.44 per ft. advance, 6 × 8 drift 9.78 per ton mined
Stoping	3204.29	1452.95		4657.23		1.388	1.432 per ton stoped
Hand tramming	1197.43	6.53		1203.97		0.358	0.350 per ton trammed
Tunnel haulage	534.95	11.83		546.78		0.163	0.162 per ton power trammed
Aerial tram	417.32	51.58		468.91		0.139	0.138 per ton trammed
Compressed air	277.59	46.41	$950.00	1274.00		0.382	0.362 per ton of break
Maintenance and repair	822.42	269.97		1092.39		0.325	0.87 per ton hoisted { prospect winze only
Hoisting	447.85	36.75		484.60		0.145	
Pumping	195.56	24.71		220.27		0.066	0.063 per ton of break
Tracks and ditches	112.75	71.06		183.81		0.055	0.053 per ton of break
Air and water lines	110.66	110.95		221.61		0.066	0.063 per ton of break
Assaying	195.68	41.61		237.29		0.071	0.068 per ton of break
Sampling and surveying	238.67			238.67		0.071	0.068 per ton of break
Steel and steel sharpening	200.93	259.76		460.69		0.138	0.131 per ton of break
Drills and drill repairs	21.69	221.15		242.84		0.072	0.069 per ton of break
Handling supplies	183.21	22.02		205.89		0.059	
Miscellaneous surface	411.80	22.95		434.75		0.130	
Supervision	791.50			791.50		0.238	
Total mining cost per ton milled						$4.624	
Ratio—$\frac{\text{Tons mined}}{\text{Tons milled}}$		1.048					

PRODUCTION COST OF ONE OUNCE OF GOLD

Tons milled	3357.
Average grade	0.33 oz. Au ton.
Ounces gold in ore	1113.3.
Total cost (mining, milling, general expense)	$24,567.79
Cost of producing one ounce of gold	$\frac{24,567.79}{1113.3}$ = $22.07

Cost Data.—Some average unit costs at the Ymir Yankee Girl property are shown herewith:

It will be noted that the "break-down" of cost varies somewhat from standard practice. Each mine has its own individuality and costs should be shown in a manner detailed to point out particular phases relating to local characteristics. The data shown in Table 1 are self-explanatory. Reference is made only to tramming costs, which may be regarded as high. However, where the ratio of vein width to lateral and vertical extent is small, such costs are usually found to be proportionately higher.

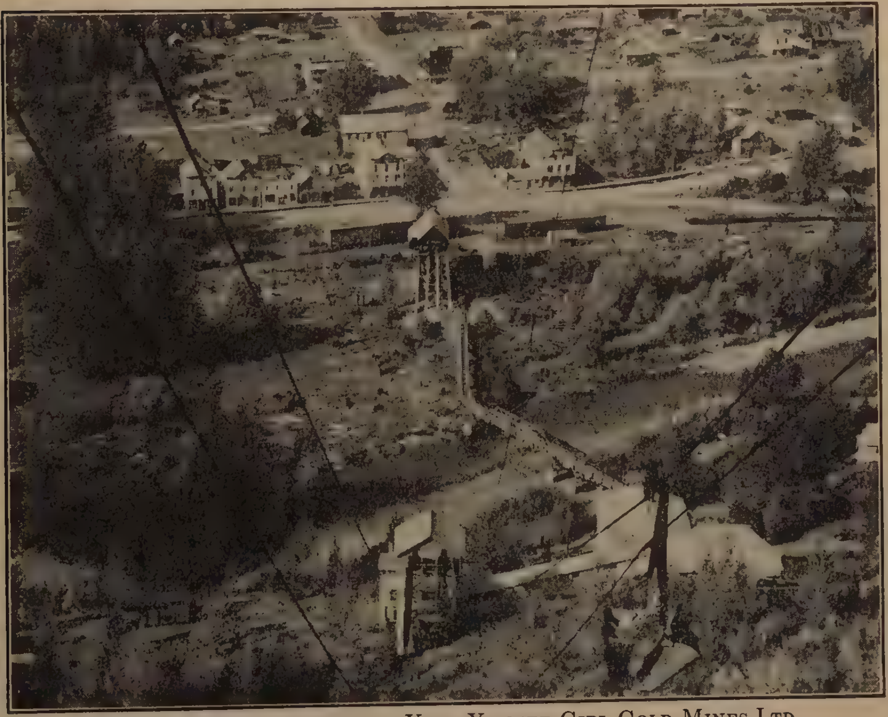


FIG. 10.—TRAM TERMINAL AT YMIR YANKEE GIRL GOLD MINES LTD.

The cost of producing an ounce of gold is an average, as are the other costs, taken over three representative months early in 1937. Of the three factors controlling cost—mining, milling and general expense—only milling is a variable with the number of metals to be produced. In the Yankee Girl mill flow sheet (Fig. 11), the separation of silver and lead add but little to the cost, and in calculating the per ounce cost of gold, this increased cost was disregarded.

However, the presence of these two metals increases the gross value of the mill heads. On the other hand, freight and smelting charges on the concentrates for this period slightly exceed the value of silver and lead shipped. The cost of smelting the gold is paid for by the silver and lead. The production cost per ounce of gold is calculated on the assumption that if the mill recovered only gold the milling costs would be no higher than with the present flowsheet, although different treatment would be required.

Since the value of the mine product is approximately 90 per cent due to the gold contained, the mine is essentially a gold mine and the mining costs are in effect costs of producing gold only.

Costs are affected by the physical characteristics of the property and distribution of workings in the mine. When taking into account the large vertical range of operations as compared with tonnage treated, the mining costs are regarded as reasonable except perhaps for the high tramming (ore gathering) cost. Total mining operation costs, including supervision, is about $4.60 per ton, of which development accounts for $0.76, stoping $1.39, supervision $0.24 and other mine expenses $2.21 per ton. The latter figure includes tramming, pumping, hoisting, supplies, maintenance, air, steel-sharpening and surveying. The average cost per ton milled is $7.03 calculated over an 18-month period, and the cost per ounce of gold produced $22.07, calculated over a recent three-month period.

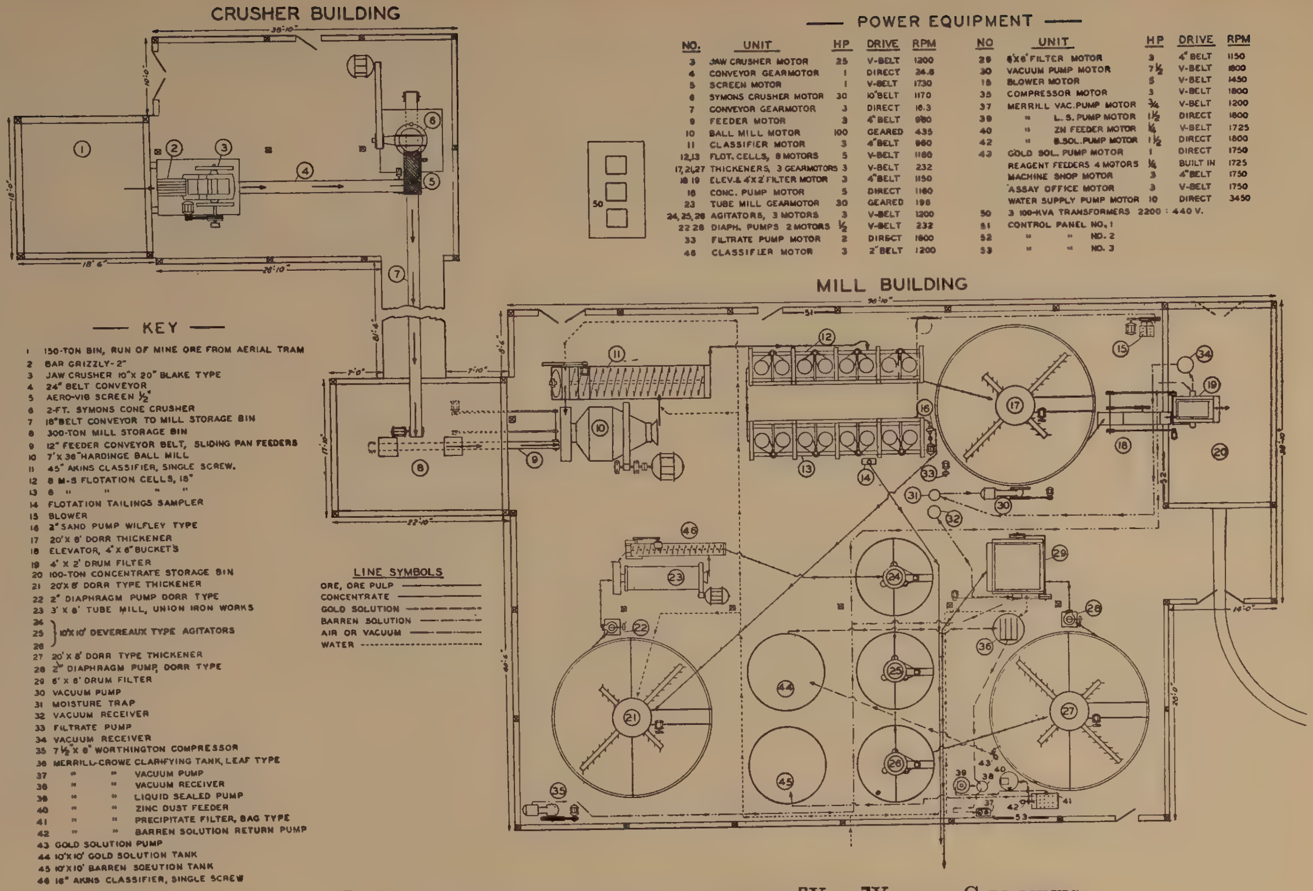
MILLING

The mill was built during the latter part of 1934. The flowsheet is shown in Fig. 11. Originally designed to treat 100 tons, the actual capacity has exceeded this by 45 tons on a number of occasions. The normal daily capacity at the present time is 120 tons. Because of the complex nature of the ore, a combination of flotation and cyanidation was finally selected. Briefly, the procedure consists in making two concentrates, a lead concentrate that is shipped to the smelter at Trail and a zinc-iron concentrate, which is cyanided. The tailing from the combined flotation concentration is discarded.

Water for the milling operation is brought from the headwaters of Bear Creek, a distance of 2 miles in 10-in. wood-stave and 4-in. iron pipe.

Electric power is purchased from the West Kootenay Light and Power Company.

Crushing.—All ore is delivered by tram buckets to the 150-ton primary ore bin. A hand-operated chute draws the ore over a 2-in. bar grizzly into a 10 by 20-in. Blake-type jaw crusher. Crusher feed is limited to 8-in. size by grizzlys throughout the ore-pass system at the mine and at the tram-loading bin. The crusher is set to deliver 2-in. discharge. A 24-in. rubber-surfaced belt conveyor carries the grizzly undersize and crusher discharge to an Aero-Vib vibrating screen of ½-in. mesh. The oversize from the screen receives secondary crushing in a 2-ft. Symons cone crusher, which reduces it to ½ in. Screen undersize



— POWER EQUIPMENT —

NO.	UNIT	HP	DRIVE	RPM
3	JAW CRUSHER MOTOR	25	V-BELT	1200
4	CONVEYOR GEARMOTOR	1	DIRECT	24.8
5	SCREEN MOTOR	1	V-BELT	1730
6	SYMONS CRUSHER MOTOR	30	10" BELT	1170
7	CONVEYOR GEARMOTOR	3	DIRECT	16.3
9	FEEDER MOTOR	3	4" BELT	980
10	BALL MILL MOTOR	100	GEARED	435
11	CLASSIFIER MOTOR	3	4" BELT	960
12,13	FLOT. CELLS, 8 MOTORS	5	V-BELT	1160
17,21,27	THICKENERS, 3 GEARMOTORS	3	V-BELT	232
18 19	ELEV. & 4'X2' FILTER MOTOR	3	4" BELT	1150
16	CONC. PUMP MOTOR	5	DIRECT	1160
23	TUBE MILL GEARMOTOR	30	GEARED	198
24, 25, 26	AGITATORS, 3 MOTORS	3	V-BELT	1200
22 28	DIAPH. PUMPS 2 MOTORS	½	V-BELT	232
33	FILTRATE PUMP MOTOR	2	DIRECT	1800
46	CLASSIFIER MOTOR	3	2" BELT	1200
29	6'X6' FILTER MOTOR	3	4" BELT	1150
30	VACUUM PUMP MOTOR	7½	V-BELT	1800
15	BLOWER MOTOR	5	V-BELT	1450
35	COMPRESSOR MOTOR	3	V-BELT	1800
37	MERRILL VAC. PUMP MOTOR	¾	V-BELT	1200
39	" L. S. PUMP MOTOR	1½	DIRECT	1800
40	" ZN FEEDER MOTOR	¼	V-BELT	1725
42	" B. SOL. PUMP MOTOR	1½	DIRECT	1800
43	GOLD SOL. PUMP MOTOR	1	DIRECT	1750
	REAGENT FEEDERS 4 MOTORS	¼	BUILT IN	1725
	MACHINE SHOP MOTOR	3	4" BELT	1750
	ASSAY OFFICE MOTOR	3	V-BELT	1750
	WATER SUPPLY PUMP MOTOR	10	DIRECT	3450
50	3 100-KVA TRANSFORMERS 2200 : 440 V.			
51	CONTROL PANEL NO. 1			
52	" " NO. 2			
53	" " NO. 3			

— KEY —

1 150-TON BIN, RUN OF MINE ORE FROM AERIAL TRAM
2 BAR GRIZZLY-2"
3 JAW CRUSHER 10"X 20" BLAKE TYPE
4 24" BELT CONVEYOR
5 AERO-VIB SCREEN ½"
6 2-FT. SYMONS CONE CRUSHER
7 18" BELT CONVEYOR TO MILL STORAGE BIN
8 300-TON MILL STORAGE BIN
9 12" FEEDER CONVEYOR BELT, SLIDING PAN FEEDERS
10 7'X 36" HARDINGE BALL MILL
11 45" AKINS CLASSIFIER, SINGLE SCREW.
12 8 M-S FLOTATION CELLS, 18"
13 8 " " " "
14 FLOTATION TAILINGS SAMPLER
15 BLOWER
16 2" SAND PUMP WILFLEY TYPE
17 20'X 8' DORR THICKENER
18 ELEVATOR, 4"X 6" BUCKETS
19 4' X 2' DRUM FILTER
20 100-TON CONCENTRATE STORAGE BIN
21 20'X 8' DORR TYPE THICKENER
22 2" DIAPHRAGM PUMP DORR TYPE
23 3' X 8' TUBE MILL, UNION IRON WORKS
24
25 } 10'X 10' DEVEREAUX TYPE AGITATORS
26
27 20' X 8' DORR TYPE THICKENER
28 2" DIAPHRAGM PUMP, DORR TYPE
29 6' X 6' DRUM FILTER
30 VACUUM PUMP
31 MOISTURE TRAP
32 VACUUM RECEIVER
33 FILTRATE PUMP
34 VACUUM RECEIVER
35 7½" X 8" WORTHINGTON COMPRESSOR
36 MERRILL-CROWE CLARIFYING TANK, LEAF TYPE
37 " " VACUUM PUMP
38 " " VACUUM RECEIVER
39 " " LIQUID SEALED PUMP
40 " " ZINC DUST FEEDER
41 " " PRECIPITATE FILTER, BAG TYPE
42 " " BARREN SOLUTION RETURN PUMP
43 GOLD SOLUTION PUMP
44 10'X 10' GOLD SOLUTION TANK
45 10'X 10' BARREN SOLUTION TANK
46 16" AKINS CLASSIFIER, SINGLE SCREW

Fig. 11.—Plan flowsheet, crusher plant and mill, [Ymir] Yankee Girl mines.

joins the cone-crusher discharge on an 18-in. conveyor belt, which carries it 100 ft. up a 16° incline to the 300-ton capacity mill-feed bin (Fig. 11).

Grinding.—Zeigler Pan-type feeders regulate the flow of crushed ore from hoppers in the bottom of the mill bin to a 12-in. conveyor belt that feeds the ball mill. The latter unit consists of 7 ft. by 36-in. Hardinge ball mill in which 4-in. steel balls are used. A 45-in. Akins single-screw classifier is operated in closed circuit with the ball mill. The classifier overflow is maintained at a density of 1.25, which is equivalent to 30 per cent solids. Screen analysis shows: plus 60 mesh, nil.; plus 100 mesh, 6.3 per cent; plus 200 mesh, 27.2; minus 200 mesh, 66.5.

TABLE 2.—*Elevations*

	FEET ABOVE SEA LEVEL
Great Northern station at Ymir	2393
Wildhorse tunnel portal	2874
Yankee Girl mine:	
1500-ft. level	3421
1360-ft. level	3544
1235-ft. level	3649
1035-ft. level	3832
935-ft. level	3925
800-ft. level	4050
620-ft. sublevel	4209
540-ft. level	4287
400-ft. level	4411
325-ft. level (Hobson)	4482
310-ft. level (Hobson)	4488
250-ft. sublevel	4555[a]
230-ft. level (Hobson)	4566
160-ft. level (Hobson)	4654
50-ft. level (Hobson)	4730[b]
Surface outcrop	4780
Dundee tunnel portal	3264
Dundee shaft (collar)	4175
Wesko camp (300-ft.) level	3689
Wesko lower (470-ft.) level	3421
Two Star adit	4200[a]

[a] Approximate.

[b] Also called "Overland Tunnel."

Flotation.—The classifier overflow is passed over corduroy blankets to extract free gold accompanying the small quantity of oxidized ore, and is carried in launders to the flotation machines. The equipment consists of two banks of eight 18-in. Minerals Separation Sub-A cells. In each bank, six cells are used as roughers and two as cleaners. Impellor shafts are driven in pairs by V-belts and electric motors. The first eight cells float a lead concentrate and, to a degree, depress zinc and iron. The tailing product, containing the zinc and iron, is then treated in the second eight cells in which all sulphides are extracted as cleanly as possible. The tailing from the second machine is rejected at this point. Flotation reagents used are:

Soda ash	2.40 lb. per ton treated
Cresylic acid	0.010 gal. per ton treated
Xanthate (pot)	0.14 lb. per ton treated
Barret No. 4	0.013 gal. per ton treated
Aero-float	0.008 gal. per ton treated

Ratios of concentration vary considerably from day to day with the changing nature of the ore. Over a period of several months, 1000 tons of ore treated has produced 79 tons of lead concentrate and 95 tons of zinc-iron concentrate.

The lead concentrate, which contains approximately 90 per cent of the gold and silver recovered by the mill, is carried by gravity to a 20 by 8-ft. Dorr-type thickener. The thickened concentrate is then raised in a bucket elevator and dried to about 9 per cent moisture with a 4 by 2-ft. drum filter. The cake is dropped directly into the 100-ton concentrate storage bin.

FIG. 12.—YMIR YANKEE GIRL MILL.

The zinc-iron concentrate is pumped by a 2-in. Wilfley sand pump to a 20 by 8-ft. Dorr-type thickener at the head end of the mill. The purpose of this thickening is to wash flotation reagents out of the concentrate in preparation for cyanide treatment.

Cyaniding.—The washed and thickened zinc-iron concentrate is pumped to the tube mill by a 2-in. diaphragm pump. The tube mill is small, 3 by 8 ft., and is in closed circuit with a 16-in. Akins single-screw classifier. The grinding medium is 2-in. flint pebbles. Screen analysis of the overflow is: plus 100-mesh, 0.1 per cent; plus 200-mesh, 12.1; minus 200-mesh, 87.8.

Agitation.—Overflow from the 16-in. Akins classifier is conveyed by launder to three 10 by 10-ft. Devereux-type agitators that are arranged in series. The three agitators treat the pulp for 24 hr. The consistency is of 1.40 density or 38 per cent solids. Reagents used are: cyanide, 8.0 lb. per ton treated; lime, 4.5 lb. per ton treated.

After agitation the pulp overflows to a 20 by 8-ft. thickener, from which the sediment is pumped by a 2-in. diaphragm pump to a 6 by 6-ft. drum filter. The cake is washed, repulped and run by gravity to the tailing pond. The thickener overflow is pumped to a 10 by 10-ft. gold-solution storage tank. From this point, the solution is filtered and precipitated with zinc dust by the Merrill-Crowe precipitation system using a bag-type precipitate filter. Reagents used in precipitation are: zinc dust, 0.45 lb. per ton concentrate treated; lead acetate, 0.90 lb. per ton concentrate treated.

Mill Sampling.—All sampling in the mill is done by hand, at hourly intervals. The method consists of cutting an entire stream of dry ore, pulp or solution, whichever the sample may be, for a duration of 18 sec. Twenty-four such cuts when combined thus not only provide a representative sample for assay but also indicate the tonnage treated. One-tenth the weight of the sample in pounds is equivalent to tons of dry ore, pulp or solution treated during the 24 hr. By recording the volume of the combined cuts, in the case of pulp, and knowing specific gravity of the dry material, the density and percentage of moisture and solids are readily determined.

ACKNOWLEDGMENTS

The preparation of this paper has been made possible by the permission to publish and the cordial cooperation of the company's officials and staff. We are especially indebted to H. W. Seamon, Managing Director, who has placed the records at our disposal and given material assistance. We are also indebted to Mr. D. M. McKinnon and the University of British Columbia for the use of polished-surface data, from which we have freely drawn. Our general geological information has been drawn largely from the work of C. W. Drysdale, J. F. Walker and W. E. Cockfield.

Prospecting, Mining and Washing the Brown Iron Ores of Alabama

By Charles Morgan*

(Birmingham Meeting, April, 1937)

An increased demand for brown iron ore in the Birmingham district during the past 18 months has caused renewed activity both in prospecting and mining these ores. In recent years the production in Alabama has ranged from 41,559 tons in 1932 to 639,351 tons in 1928. During the last 9 years about 83 per cent of the total brown ore produced in the United States has come from Alabama (Tables 1 and 2).

Prospecting

Two methods of prospecting are used in this district: (1) test pits, (2) churn-drilling. Usually there is certain quite definite information on an area to be prospected; some of the indications are float ore such as small gravel or small and large boulders known as "dornicks" scattered over the surface, and ore exposed by erosion in gulleys and ravines, which have cut or eroded through the overburden. Often these natural erosions furnish an indication of the thickness of the ore and to some extent of the area of the ore body.

Deposits of brown ore are very irregular both in thickness and extent. Deposits are often lens-shaped or long narrow leads, some of which may be parallel and separated from one another by sand, clay or gravel. The bottom—the footwall—is also very irregular. This footwall is usually lime clay—disintegrated limestone, locally called "white horse," which usually is found at a depth of 6 in. to 20 ft. overlying the limestone. At some places sand and gravel is found under the ore instead of clay. No ore is found under the lime clay, although often the ore has sunk or slumped and is found in the upper portion of this clay, as at some of the Sloss workings at Russellville.

The origin of these ores has been discussed by Major E. C. Eckel and Dr. E. F. Burchard in U. S. Geological Survey *Bulletin* 400, and recently the latter has done further work on ores in the Russellville district, of which reports undoubtedly will be published.

Manuscript received at the office of the Institute April 7, 1937. Issued as T.P. 860 in Mining Technology, November 1937.

* Consulting Mining Engineer, Bessemer, Ala.

In test pitting, circular pits are dug, from 30 to 34 in. in diameter. In method 1, the area to be tested is blocked off in 200-ft. squares and holes are dug 200 ft. apart. If ore is found, this method is continued, a hole often being sunk in the center of the square. If ore is not found in all holes, other holes are dug in the spaces between those first made, and the holes showing ore thus fairly definitely outline the ore body. Method 2 is to sink a hole where surface and other indications indicate possible ore and follow the lead or strike of the ore with holes spaced from 100 to 200 ft. apart, then to sink holes to determine the width of the ore body.

TABLE 1.—*Production of Brown Iron Ore from* 1928 *to* 1936[a]

Year	Alabama[b]	United States
1928	639,351	816,637
1929	626,163	775,610
1930	587,389	720,496
1931	245,663	359,960
1932	41,559	61,739
1933	207,981	235,297
1934	277,904	286,073
1935	249,241	267,183
1936	440,746	474,889

[a] U. S. Bureau of Mines.

[b] Approximate average of 48.66 per cent Fe natural and 50.7 per cent Fe dried.

TABLE 2.—*Production of Brown Iron Ore in Alabama by Counties in* 1935 *and* 1936[a]

County	1935		1936	
	Number of Mines	Gross Tons	Number of Mines	Gross Tons
Bibb and Tuscaloosa	3	49,741	9	137,055
Blount	2	102,204	2	108,254
Butler and Crenshaw	3	10,960	3	3,701
Calhoun			4	23,597
Cherokee	1	334	4	5,100
Chilton	1	310	3	15,303
Cleburne			1	948
Coosa	1	11,527	1	5,370
Franklin	1	49,507	2	114,876
Shelby	1	20,327	2	20,326
Talladega	4	4,331	6	6,216
Total	17	249,241	37	440,746

[a] U. S. Bureau of Mines.

Method 1 was used in the Woodstock district, where the deposits were fairly uniform; method 2 was used in the Russellville, eastern Alabama and Talladega districts, where many of the deposits are irregular and of smaller extent. If deposits are very irregular, in using method 2 close prospecting is necessary, and pits should be spaced not more than 100 ft. apart.

The circular pits are dug by hand. Two men constitute a crew (Fig. 1). Tools and equipment required are two short-handled shovels (handles not over 12 to 14 in.); four or five picks with 16 to 18-in. handles, with one end cut off and one clay pick point; two or three gads; 50 ft. of 1-in. manila rope; one windlass and frame; one bucket, made from a nail keg with straps of iron across the bottom and up the sides, and a loop in the bail. The windlass is about 6 ft. long, 6 to 7 in. in diameter; in one end of the windlass is driven a 1-in. rod and the other end has a ¾-in. iron crank. One end of the rope has a pigtail hook, which hooks into the loop of the bucket bail as a safety measure.

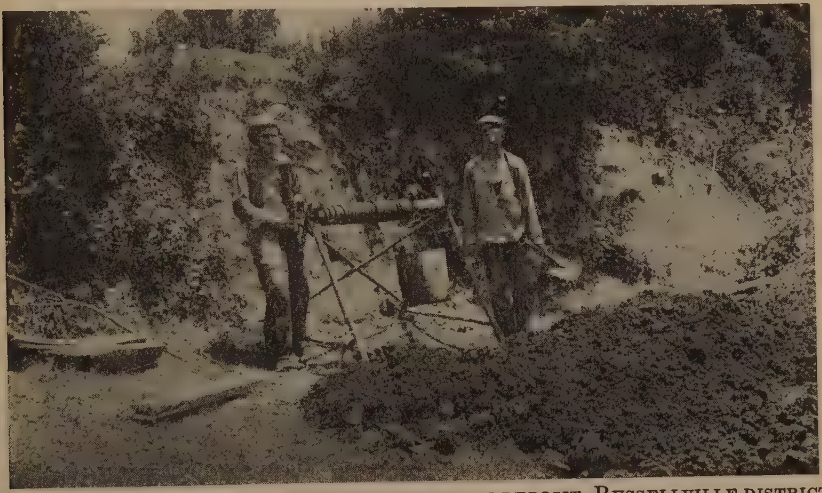


FIG. 1.—SINKING PROSPECT PITS IN BROWN IRON-ORE DEPOSIT, RUSSELLVILLE DISTRICT, ALABAMA.

The frame or support for the windlass is made from 1 by 4-in. lumber or small poles (Fig. 1). Fishtail-pointed bars are also used for loosening the material, instead of picks. Experience is needed for digging the holes and usually in each brown-ore district there are men that make a business of this work and are proficient in it. The material is dug out with a shovel to a depth of from 6 to 7 ft., then the windlass and frame are set up over the hole and one man digs while the other draws up the material. The different kinds of material as dug are dumped in separate piles for later inspection. These pits can be dug 50 ft. deep, or more, depending upon whether or not water appears.

Circular holes stand up better and have less tendency to cave than rectangular holes. In sinking through sandy material, a method of curbing devised by the writer has been used with success. This consists of bands of ½ by 2-in. iron of the approximate circumference of the hole, with 5-ft. pieces of 24-in. corrugated roofing for curbing. These are easily inserted and held in place by wedges, also by 40-dwt. nails through holes in the bands into the side wall. This curbing takes up very little space and can be recovered when the hole is finished.

Prices for digging range from 25¢ to 50¢ per foot, depending on material and depth.

METHOD OF SAMPLING

Method 1.—A vertical trench 2 to 3 in. deep is cut through the ore section after the pit has been finished. The volume of this sample is measured, then washed and weighed, from which the amount of ore per cubic yard of material is computed. This sample is then cut down and a portion saved for analysis.

Method 2.—All the ore removed from the hole after breaking dornicks and lumps is quartered down to a sample suitable for washing as in method 1. The yield is computed from the diameter of the hole and the thickness of the ore. This is the more accurate method but requires more time than the other method and is more expensive.

Advantages of method 1 are that material can easily be seen, it is cheaper than method 2 for shallow holes, several crews can be worked, equipment is not as expensive, and it is suitable for preliminary work.

CHURN-DRILLING

Often in a test pit hard ore is struck at a depth of 25 to 30 ft. Blasting often causes the holes to cave and makes it impossible to ascertain the thickness of ore. Often, also, water prevents deeper sinking of the holes. The Republic Iron and Steel Co. was the first to prospect on an extensive scale with a churn drill in material of this kind, in 1909 at the old Houston mine, Reno, Tuscaloosa County. Satisfactory results were obtained, as was shown by later mining operations.

In churn-drilling two methods are used: (1) starting from the surface, drilling and casing to limestone; (2) drilling in the bottom of test pits that have been sunk as deep as possible. In both methods casing of drill holes is essential; 6-in. black pipe is commonly used, although flush-joint 5⅝-in. casing has been used. A No. 3 Keystone tractor well drill is the machine most commonly used, driven by gas or steam.

Because of the tendency of the ore to ravel or cave off, it is necessary to drive casing after drilling 1 ft., then bail out material, saving some in tubs or barrels for samples. Erroneous results from churn-drilling are likely to be obtained unless holes are cased in this way. The cuttings from the churn drill when in ore are saved in barrels, washed and sampled, and the volume is computed to check with the diameter of the hole.

An experienced operator with careful engineering supervision obtains reliable results. The cost of churn-drilling varies from 50¢ to 75¢ per foot. The casing is usually recovered from the holes. One advantage is speed for deep-hole, and testing of areas that otherwise could not be prospected by test pits.

MINING

The districts in which brown ore has been mined in Alabama are: near Woodstock, Tuscaloosa County; Champion and Taits Gap, Blount County; Tecumseh, Baker Hill, and Rock Run, Cherokee County; Ironaton district, Talladega County; vicinity of Anniston, Calhoun County; Shelby, Shelby County; Brierfield, Bibb County; a few points in Chilton, Butler, and Crenshaw Counties, and near Russellville, Franklin County. The Woodstock, Champion, Butler County, Russellville, and Talladega districts are now the principal producers of brown ore. When an ore area has been developed by prospecting as outlined, the larger operators usually have a surface contour map made of the area, to facilitate the location of washing plants, muck and water ponds, railroads and buildings. An ample water supply should be secured and ample storage space should be provided for setting and clearing muck. It is usually necessary to settle and clear the water before turning it into creeks or streams; also, it may be necessary to re-use some of this water.

Brown ore is mined from open cuts. Steam shovels of various makes are used, ranging in size from 20 to 80 tons. Haulage is usually by narrow-gauge (36-in.) railroads. Steam locomotives from 16 to 24 tons are used for haulage, with side-dump cars of 4 to 6 cu. yd. capacity. Where stripping is necessary, usually the same equipment is used, although one mine had a 150-ton shovel working on the same principle that is followed in strip coal mines—that is, opening a cut or area, removing the ore and casting the overburden into the mined-out area. It is the author's opinion that a dragline shovel could be used effectively in many of the stripping operations. Caterpillar shovels are now used and are a great improvement over the old railroad type, as they are easier to move and clean the areas better.

A small tonnage is also produced yearly by screening or dry mining, chiefly in isolated deposits and old workings. Very little equipment is necessary for such mining—a few picks and shovels, one screen with 1-in. mesh and one with ¾-in. mesh. Material is dug, larger lumps picked out by hand and other material cast against an inclined screen. Coarse ore is saved and dirt, sand, and considerable fine ore are wasted. The ore is trucked to the nearest railroad—a distance of 12 miles in one area.

In the mining of brown ore, considerable care is required because of the irregular size of the deposits and the impurities in the ore. Often there are barren pockets and lenses of clay, gravel and sand, and it is better practice to waste or discard this foreign material than to let it go to the washer, even if a small amount of ore is wasted with it.

Yields or recovery of washed ore per cubic yard of material in an ore body vary greatly in the different districts as well as in each district, and vary from day to day. A high recovery is 1 ton of ore to 1 cu.yd. of material fed to the washer, and not many deposits of any size give this recovery. An average recovery where no stripping is necessary is 1 ton to 2½ or 3 cu. yd. Where stripping is necessary this may be as high as 7 cu. yd. for 1 ton of ore.

The thickness of stripping that can be removed depends on the thickness of the ore, although at times it may be necessary to strip small areas that carry very little ore but which are adjacent to a larger ore body that can be mined to better advantage if the small areas are stripped.

A recent change in mining has been brought about by the use of trucks instead of tramcars and locomotives for transportation of ore from shovel to washer and from washer to railroad cars. Several deposits ranging in size from 150 tons or less to several thousands of tons have been mined by this method at the Wade and Richey washer, near the old Huston mine, Tuscaloosa County. The ore is mined by one 31 type shovel and one 50 type, loading into trucks, 2½ to 3 cu. yd. capacity, equipped with hydraulic dumps, and is trucked about 1½ miles to a two-log washer. The trucks back up an inclined ramp and dump directly onto the grizzlies. Twelve to fifteen trucks are used and hauling is done by the cubic yard. This is a typical two-log washer, with grizzlies, cone screen over logs, roll crusher, two 30-ft. steel logs, double-shell sand screen and picking belt, the ore discharges into the railroad cars.

Electric power is used at the washer and for pumping. Three hundred tons is an average daily run, although 550 tons has been washed at this plant in 10 hr. This compares favorably with plants using cars and locomotives for haulage, and has the following advantages: Small deposits and areas in abandoned workings can be worked, which would not justify the building of a tram road, and the cost of equipment is low, as trucks can be rented. Comparatively rich ore, or ore in which recovery is about 1 ton ore to 2 cu. yd. of material, can be mined successfully by this method, but on leaner material in which recovery is 1 ton ore to 3 cu. yd. or over, and on longer hauls—over 2 miles—and on large deposits cars and locomotives would probably be the cheaper.

Maintenance of roads in the continuous rainy season during the winter months must be considered, but costs very little more than the constant shifting of tracks for locomotive haulage. Usually either gravel or chert is available for road surfacing.

WASHING

A typical flowsheet of a brown-ore washer is given in Fig. 7. There are many variations of this. Finer crushing requirements recently have caused additional installation at some washers of a secondary crushing to a maximum of 2 in.; either a roll crusher or a Simons cone crusher is used.

In the early days of washing, there were many makes and types of ore washers, but this machinery has been pretty well standardized at present (Figs. 2 to 5).

In certain districts jigging the ore below ¾-mesh is also necessary because the overburden consists of gravel or chert or because there are gravel and chert in the ore deposit. "All jig material is sized before it is sent to the jigs, so that each cell operates on a uniform size. All material is jigged under water. This means a much more efficient operation. The waste, flint and gravel are removed to a bin by one elevator and the finished ore to another bin by the second elevator. All clay should be removed from the feed to the jigs. Each four-cell jig will handle 20 to 25 tons of washed flint and ore per hour. An average would be approximately 100 tons of clean ore per four-cell jig per day, operating at 75 r.p.m. Jigs give best service when operating on sizes up to minus ¾ in. The jig is generally equipped for plus 1/16-in. minus ¼-in., plus ¼-in. minus ½-in., plus ½-in. minus ¾-in., and plus ¾-in. minus 1⅛-in." (McLanahan Stone and Mchy. Co.)

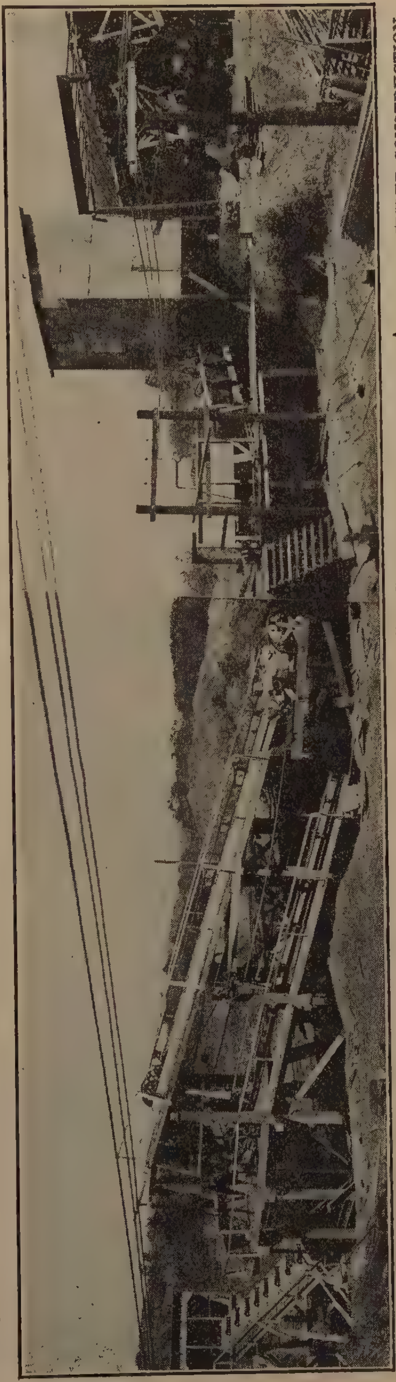


FIG. 2.—BROWN IRON-ORE WASHER OF SHOOK AND FLETCHER SUPPLY CO., RUSSELLVILLE DISTRICT, ALABAMA, UNDER CONSTRUCTION. Right to left shows grizzly, crusher house, log washer, cone screen, conveyer belt, and storage bins.

Hancock jigs were used at one time in the Woodstock district. They jigged on unsized material, plus 1/16-in. to 3/4 or 1-in. mesh, and were fairly satisfactory in removing angular chert and low-grade ore, but did not efficiently remove gravel in the Russellville district.

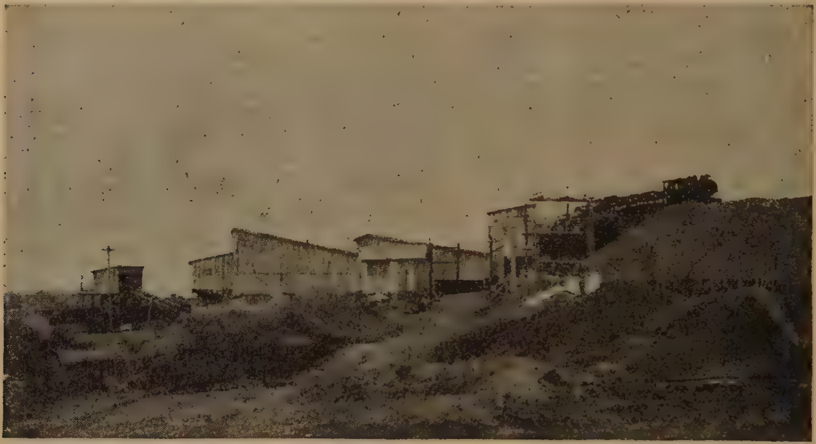


Fig. 3.—Washer shown in Fig. 2, completed and in operation.

A recent method of mining and washing small and isolated deposits of ore is by the use of revolving trommels (Fig. 6) driven by gas engines and water pumped by gas engines. The material at the ore bank is often dug by hand, loaded into trucks and hauled to the washer, and the washed ore is trucked to nearest railroad. Sometimes a small gas shovel—3/8 to 1/2-yd. dipper—is used in loading trucks. Fifty to 150 tons a day

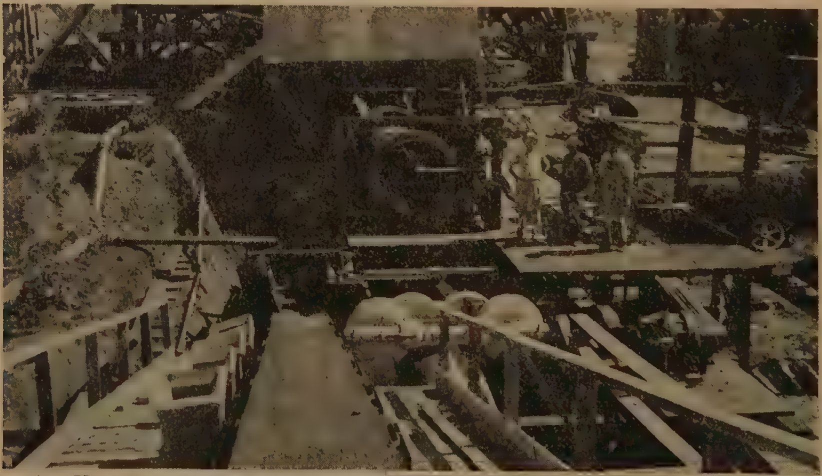


Fig. 4.—View within washer shown in Fig. 2, looking down conveyer belt into cone screen and log washer.

can be washed on this type of washer. Material from trucks is dumped onto inclined grizzlies, spaced 3½ to 4 in. Large sizes are broken by hammer to pass through the grizzlies and this with the minus 3-in. material, goes to a revolving screen, 10 to 14 ft. long, with ½ to ¾-in. openings, over which is fastened ⅛ to 3⁄16-in. mesh wire cloth. The minus 3⁄16-in. material is wasted. Spray pipes are used both inside and outside on the screen. From the screen, ore discharges into a short chute, where clay balls, sandstone and gravel are picked out. The chute discharges into a small ore bin or into a car if the plant is on a railroad. If discharged into a bin, the ore is trucked to a railroad.

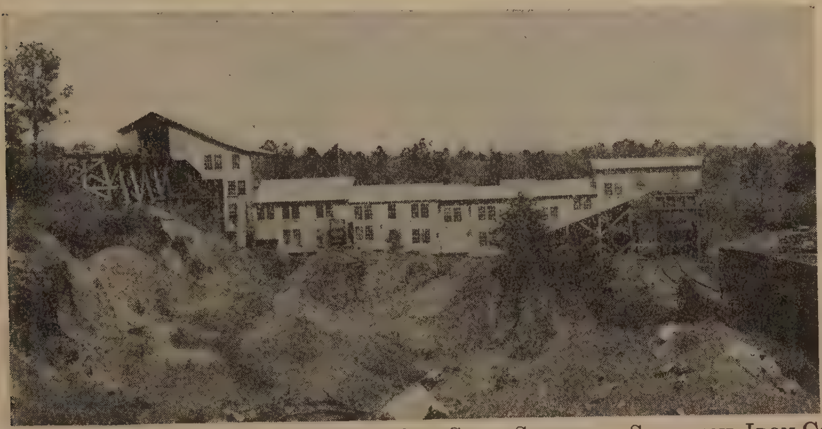


Fig. 5.—Brown iron-ore washer No. 12 of Sloss-Sheffield Steel and Iron Co., Russellville district, Alabama.

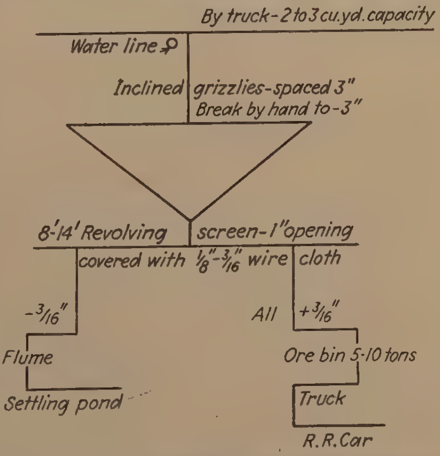

By truck-2 to 3 cu.yd. capacity
Water line
Inclined grizzlies-spaced 3"
Break by hand to -3"
8'-14' Revolving screen-1" opening
covered with ⅛"-3⁄16" wire cloth
-3⁄16"
All
+3⁄16"
Flume
Settling pond
Ore bin 5-10 tons
Truck
R.R. Car

Fig. 6.—Flowsheet with revolving screen.

Driven by 7 to 10-hp. gas motor or by electric power. Water supplied from nearest source by gas-driven centrifugal pump or by electric power.

From 150 to 350 gal. per min. of water is required, which is supplied by gas engine.

Advantages are cheap equipment and not much capital required. Many small deposits can be worked by this method and old abandoned workings reworked. Very little skilled labor is required.

Disadvantages are that only richer deposits and those not containing much clay mixed with the ore can be worked, as screens will not cut up the clay and logs are required for this class of material. The addition of a short picking belt would be an advantage.

A flowsheet of the Shook and Fletcher Supply Company's brown-ore washer is shown in Fig. 7. Raw ore is transported from the mines in trains of five cars—side-dump cars of 5 cu. yd. capacity—on track of 36-in. gauge by 20 to 25-ton steam locomotives.

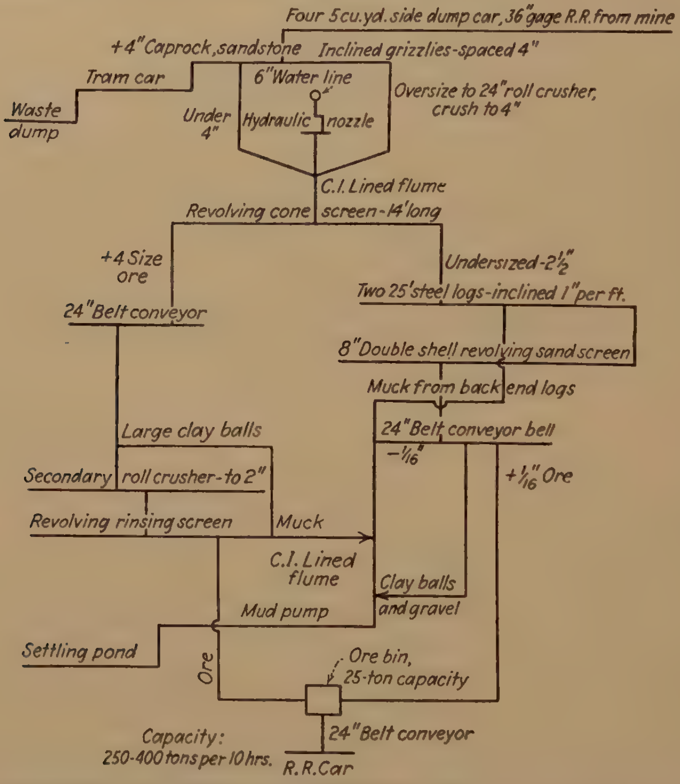


FIG. 7.—FLOWSHEET OF SHOOK AND FLETCHER SUPPLY COMPANY'S BROWN-ORE WASHER, RUSSELLVILLE, ALABAMA.

The ore is dumped on inclined grizzlies, spaced 4 in. Over 4-in. goes to single-roll, 24-in. crushers crushing to 4 in., from which it goes to a cast-iron lined flume. Material passing through the grizzlies is washed down into the flume by a nozzleman. Material from the flume discharges into a conical revolving screen, 2-in. openings first 7 ft., 3-in. openings last 7 ft. Plus 3-in. goes to 24-in. belt conveyer, where large clay balls, sandstone and conglomerated lumps are picked off by hand. The belt discharges to a secondary roll crusher, crushing to 2 in., and goes to a rinsing screen and then to an ore bin. The minus 2 and 3-in. from the revolving screen goes to two 25-ft. steel logs, inclined 1 in. per foot. These are the latest kind of logs—square plates, about 10 in.

welded together, to which angles are welded to form a double screw. Paddles are bolted to these angles and rotation of the logs agitates the ore and muck and works the mixture forward to the discharge end of the log. The log revolves about 20 r.p.m. Muck is discharged at the lower end of the logs, through an adjustable gate, to a cast-iron muck flume then to a centrifugal mud pump and by pipe line to a settling pond. The log box is built of sheet steel and driving gears are enclosed in a steel housing in which oil can be maintained at the required level.

All the ore from the logs is discharged to a revolving 8-ft. double-shell sand screen. The inner shell has ½ by 2-in. slotted openings; the outer shell is of steel with 3⁄32 to 1⁄16-in. openings. Material passing through this outer shell is mainly fine sand and some fine ore. All ore from this screen discharges to a 24-in. belt conveyer, which discharges to an ore bin. Small clay balls, gravel and sandstone are picked by hand from this belt and sent by short chutes to the mud flume. Spray pipes are used both inside and outside on both conical and sand screens. Ore from the ore bin is discharged to a belt conveyer, which carries it to the railroad car.

On the chute from the sand screen to the picking belt, a trap door can be installed so as to divert the minus ½-in. ore to a cross conveyer to jigs or sizing screens for jigs. To date these jigs have not been installed, but some of the ore that will be mined later will require jigging.

This plant is operated by electric power. For a plant of this type the following are required: one 50-hp. motor for logs and conveyers, one 25 to 30-hp. motor for primary crusher, one 25-hp. motor for secondary crusher, one 5 to 7½-hp. motor for ore belt to railroad cars, one 25 to 30-hp. motor for water supply, depending on head pumped against, one 25-hp. motor for mud pump, and one 35-hp. motor for jig plant.

There are many variations of arrangement of crushers, belts, different spacing and slope of grizzlies, flumes, and screen openings, which work to the best advantage in the several districts and on different types of ore, according to the ideas of different operators.

Washed ore weighs 112 to 118 lb. per cubic foot. Several attempts have been made to recover the minus 1⁄16-in. ore passing through the outer shell of the sand screen; some pass it over a finer screen; some use a kind of hydraulic classifier. Usually this 1⁄16-in. material carries considerable sand and lower grade ore.

TAILINGS

Prof. J. R. Cudworth has done some work at the U. S. Bureau of Mines Experiment Station, University, Ala., on tailings from log washers and old muck ponds, using tables for separation. It should be remembered that brown iron ore is a low-priced product and usually an ore body is worked out in a few years, and the price of ore does not commercially warrant the installation of expensive equipment. Some method should be worked out to recover this fine ore, as it has already been mined and washed and in places in certain districts it carries enough ore of good grade to warrant the cost of recovery. At some plants, by crude methods 3 tons is saved at certain times during a day's run—10 hr. This is sometimes rewashed, using less water in that operation, and a fair recovery is obtained. Some fine ore is also lost with muck from logs; using too little water makes a thick muck, which carries off in suspension some fine ore, while too much water increases the velocity of the muck discharge and carries off some fine ore with the muck.

Uniformity of Ore and Prices

Iron ore is sold on the unit (or per cent) of iron basis; 5½¢ per unit of iron f.o.b. furnace has been average price for 1936. Phosphorus varies in each ore district and in each deposit, often samples taken less than 50 ft. apart show a variation of 0.30 to 0.50 per cent. Low-phosphorus ore often commands a slight premium if it contains less than 0.30 per cent P. High-phosphorus ores are those above 0.50 per cent.

Ores also carry varying amounts of manganese, from 0.25 to as high as 4 per cent. At present no premium is paid for manganese, but total metallic content (iron and manganese) is paid for on unit price of iron. The insoluble material also varies.

In recent years, some of the furnace companies have begun to deduct for moisture content, which runs from 4 to 9 per cent, and ore is paid for on a dry basis; some companies allow 5 per cent moisture.

Where it is possible to stockpile ore, as has been done by Shook and Fletcher, and then later reload it, a more uniform shipping ore is obtained. When the furnace companies are receiving more ore than they consume, stockpiling is also used, securing the same results, or low-phosphorus and high-phosphorus ores can be stocked separately.

Acknowledgment

The author is indebted to Dr. E. F. Burchard, of the U. S. Geological Survey, for the photographs that illustrate this paper.

Sublevel Caving, Large-pillar Method, at the Montreal Mine

By R. A. Bowen,* Member A.I.M.E.

(New York Meeting, February, 1938)

The Montreal mine, operated by the Montreal Mining Co., is four miles west of Ironwood, Mich., in Iron County, Wisconsin. It is the westernmost producing property on the Gogebic Iron Range of the Lake Superior district. Its production is 4000 tons of hematite per day from an average depth of 2300 ft. This tonnage is hoisted through a five-compartment vertical shaft. Several mining methods are in use but in the larger orebodies the method is sublevel caving.

Ore Occurrence and Characteristics

The main orebodies of the Montreal mine lie in eastward-pitching troughs formed by the intersections of intrusive dikes with impervious members of the iron formation. The formation has an average width of 425 ft., strikes east and west and dips north at approximately 62°. The dikes dip south at 45° and bear north 20° from the strike of the formation, making the intersections with various members pitch east at 16°.

The dikes are not continuous across the formation but are interrupted by a longitudinal bedding fault, which has a throw of 380 ft. vertically and 885 ft. horizontally. North of the fault zone the formation is called "hanging," south it is called "footwall" formation.

The faulting has created two ore-bearing horizons; viz., at the intersection of hanging dikes with the slate members and the foot dikes with the quartzite footwall. In each formation the main ore-bearing member consists of wavy beds of hard chert interlaid with hematite.

In the most productive part of the mine, where the sublevel caving method is employed, a hanging-formation dike lies in juxtaposition with a footwall dike and the iron formation carries ore in certain sections from the quartzite footwall to the hanging dike. Here a horizontal section of the orebody may be 300 to 400 ft. wide for a length of 150 ft. or more and tail out lengthwise in both the foot and hanging formations. The footwall ore section is much longer than the hanging wall because of the orebody's greater height above the "pitching dike."

Manuscript received at the office of the Institute Nov. 26, 1937. Issued as T.P. 886 in Mining Technology, March 1938.

* Assistant Superintendent, The Montreal Mining Co., Montreal, Wisconsin.

The ore is classified as soft. Most of it has a claylike consistency with hard ribs throughout. It stands unsupported very well yet breaks fine when blasted. Relatively large openings, especially under new capping, may be made in it without immediate caving. However, it is thick and wavy bedded, therefore must be trimmed carefully and watched closely. The average iron content of the ore in place is 57 per cent Fe natural.

Enclosing Rocks

The ore is underlain by a strong self-supporting quartzite footwall which stands at 62°. The capping is strong and hard but has been weakened longitudinally by interlaminated seams of ore and transversely by cross-jointing planes 8 to 20 in. apart. The overlying rock or capping will stand unsupported while the ore is taken out but gradually slabs off and fills the opening caused by the extraction of the ore. The capping breaks down without extremely large blocks or much fines. On account of the dip of the formation and pitch of the dikes, new capping is exposed 80 ft. to the north and 350 ft. to the east for each 100 ft. of vertical depth. The average iron content of the marginal formation rock is 37 per cent natural. The formation rock in contact with the ore usually runs higher in iron than the average but the change from ore to rock is comparatively abrupt.

Choice of Mining Method

A sublevel caving method of mining is well adapted to the conditions prevailing. A high-grade ore occurs in a rich formation, which permits the unavoidable dilution incident to mining without destroying the commercial value of the product.

The records of extraction indicate that dilution degrades the ore from 57 per cent Fe natural in place to 54 per cent Fe natural in shipping product.

The size of the pillar attacked, 50 by 50 ft. in cross section, is determined by the inherent characteristic of the ore to stand unsupported and the time an opening may be maintained under the crushing effect of the gob. The slowness of the gob movement is one of the determining factors in making top slicing impracticable.

Development

The ore bodies in the Montreal mine are scattered along two miles of the formation and are approached by main level drifts through the quartz slates south of the footwall. The quartz slates are much easier to penetrate than the cherty formation and stand well when gunite is applied. With a drift south of the footwall, transportation from one ore body through another is avoided and fresh air may be distributed directly to the ore bodies and exhausted to the level above.

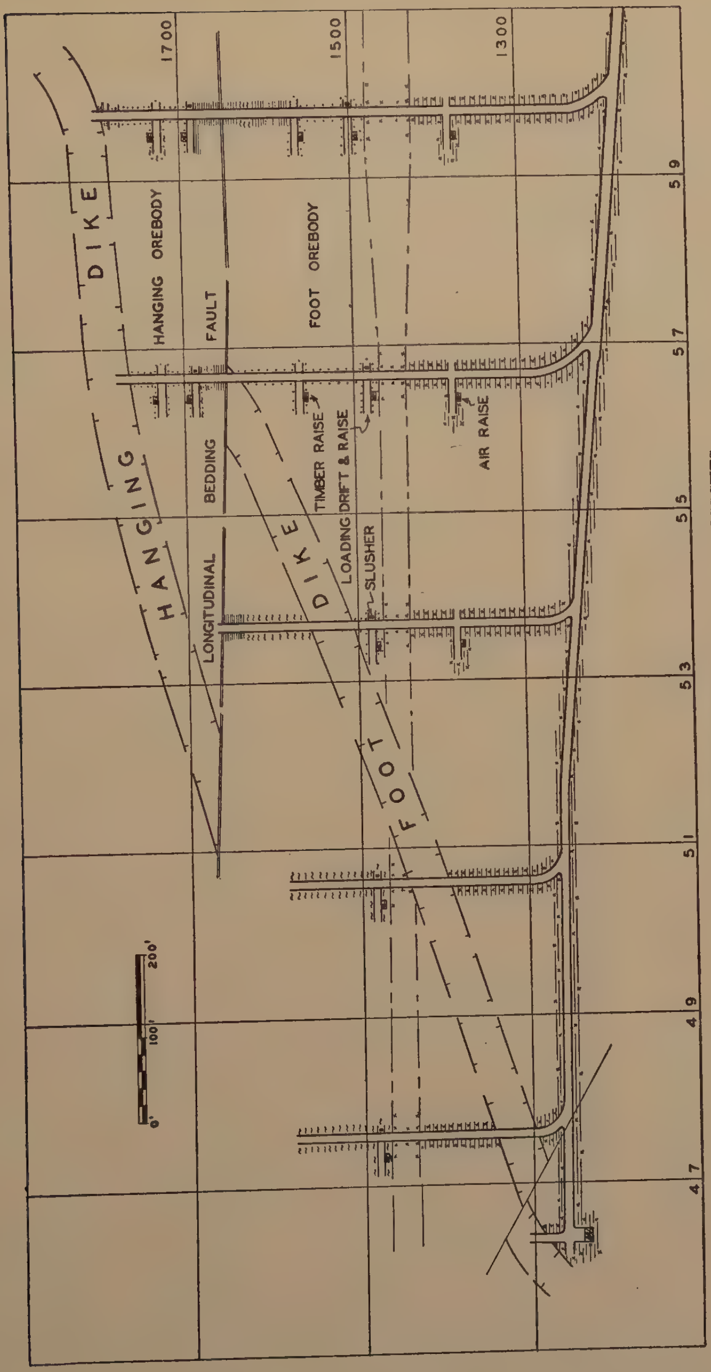


FIG. 1.—ARRANGEMENT OF LEVEL CROSSCUTS.

To open up the ore bodies, main-level crosscuts are driven north from the slate-rock haulage drift at intervals of 300 ft. "Loading drifts," from which trains are loaded, are driven in both the footwall and hanging ore bodies (Fig. 1). A double-compartment mining raise without chutes is driven from each loading drift. The foot raises are near and parallel to the quartzite footwall and the hanging raises are in the center of the ore body, parallel to the dike. North of each mining raise a second raise is driven to handle timber and materials and to serve as a travelway and airway.

The foot mining and timber raises are next connected on the mining sublevel. The connecting sublevel crosscut is then continued north to the hanging ore body and south to the ventilation raise in the quartz slates (Fig. 2). The main slushing drift is driven east or west to meet a slushing drift from a similar procedure in the adjoining crosscut. This completes the development of the 300-ft. block of ground.

The main haulage-level interval is 150 ft. and the sublevel interval is 50 ft. As the level pillars are mined directly into cars, the raises are driven only to the second sublevel, which is 100 ft. above the loading drifts.

The development is planned as described above in order to make the ore accessible with the least amount of raising, which is the most dangerous and costly of the operations. Raises are also difficult to maintain, because of timber decay. When the slusher came into use, branch raises on 25-ft. centers were abandoned and the ore was slushed north and south to raises spaced along the strike in the middle part of an ore body. When more reliable and more powerful slushers were built in easily portable units, it was found that the raises in each pillar along the strike could be dispensed with. The present practice is to slush the ore south to the footwall, then east or west along the strike to one raise. By the use of the slusher, the area served by a raise has increased from 25 by 25 ft. to 150 by 150 ft.

Breaking Ore

Breaking ore by the sublevel caving method consists of crosscutting to the limits of the ore and breaking and caving the side and back pillars in retreat. The ore is blocked out in 50-ft. pillars along the strike of the formation starting from a point 150 ft. east or west of the mining raise and retreating to the raise.

In attacking a 50-ft. pillar, two crosscuts are driven at 25-ft. centers from the main slushing drift. A manway is made by raising midway between the crosscuts and driving an untimbered subcrosscut 25 ft. above the mining sublevel (Fig. 3). Openings from the inside of each crosscut are carried up, enlarging as they progress, until they hole through to each other and catch the manway. From this point the "stope" is enlarged from narrow benches. As slice holes are blasted in the walls, the back caves down until it arches over the additional width of opening created. The operation is carried on in this manner until the protecting shell of ore around the "stope" is remarkably thin. It is usual for the gob to be exposed in several places before the "stope" begins to show weight and slab off the back. At this stage any supporting ribs are cut and the "stope" enters the "dropping" stage, which continues for several shifts. The opening made is confined within the limits of the 50-ft. pillar. Orientation is easy from the location of the manway and openings from the two crosscuts (Fig. 4).

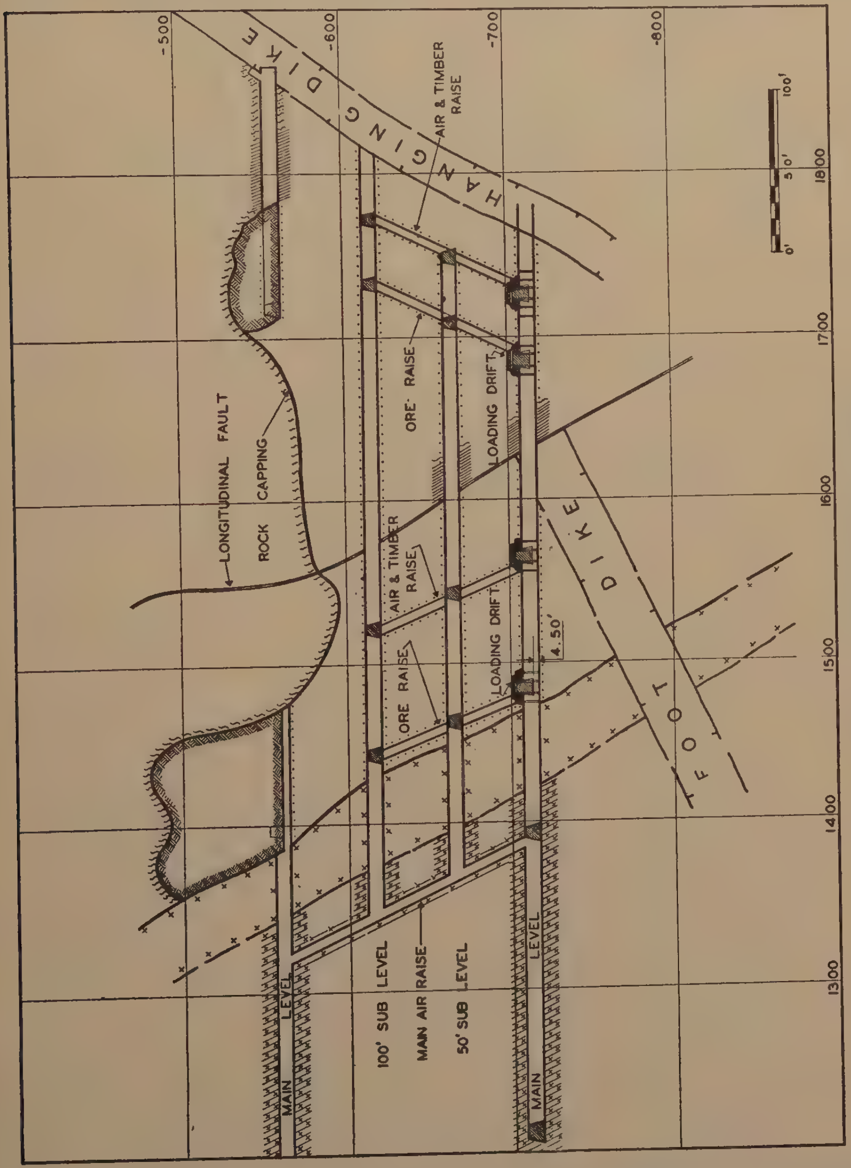


FIG. 2.—CROSS SECTION OF MINE.

After the pillar has been mined back to the slushing drift, a stope is opened on the footwall from the drift. If the footwall is faulted back, the cross-haul hoists are transferred across the drift and crosscuts driven south. Special care is taken to prop the drift for the passage of air after the mining has been completed.

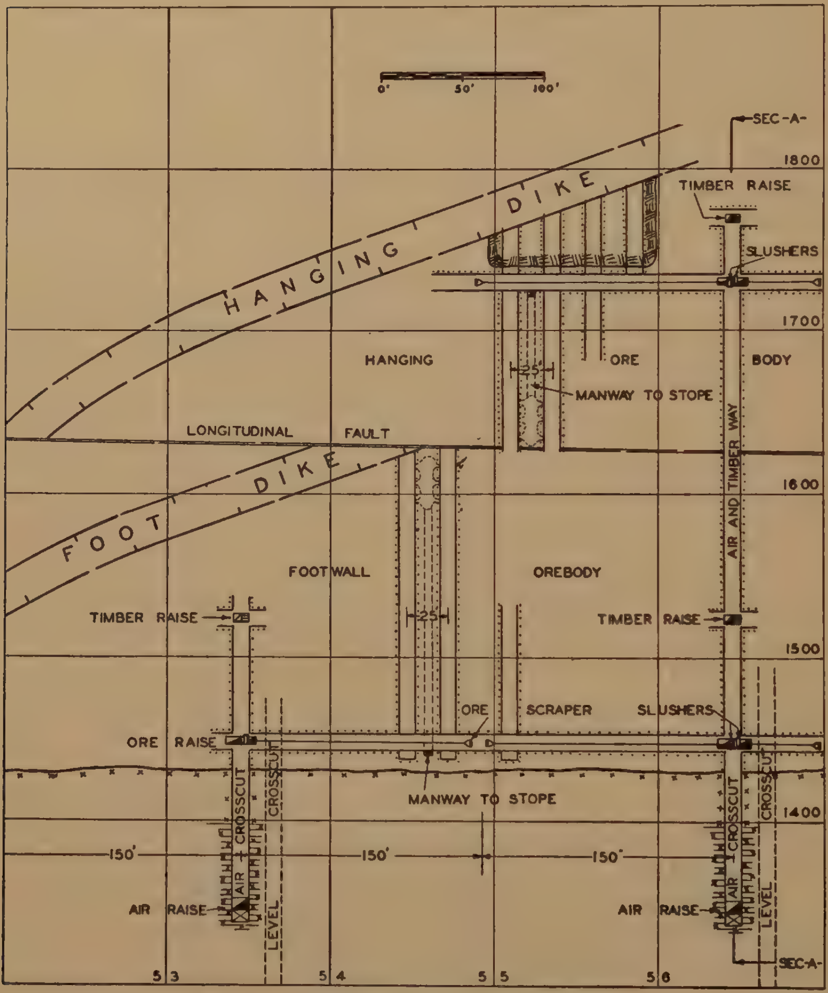


Fig. 3.—Plan of 100-foot subcrosscut.

Drilling and Blasting

The ore is drilled with 2⅝-in. wet hammer machines using ⅞-in. quarter octagon hollow drill steel with square shanks and detachable bits. Rock is drilled with 4-in. wet hammer machines using 1¼-in. hollow round lugged steel.

In the ore, 1⅛ by 8-in. 40 per cent gelatin dynamite is used and in the rock 1¼ by 8-in. 60 per cent gelatin dynamite. No. 6 blasting caps and 7-ft. double-tape fuse are used. In the sublevel operations, ⅝ lb. of 40 per cent dynamite per ton is consumed.

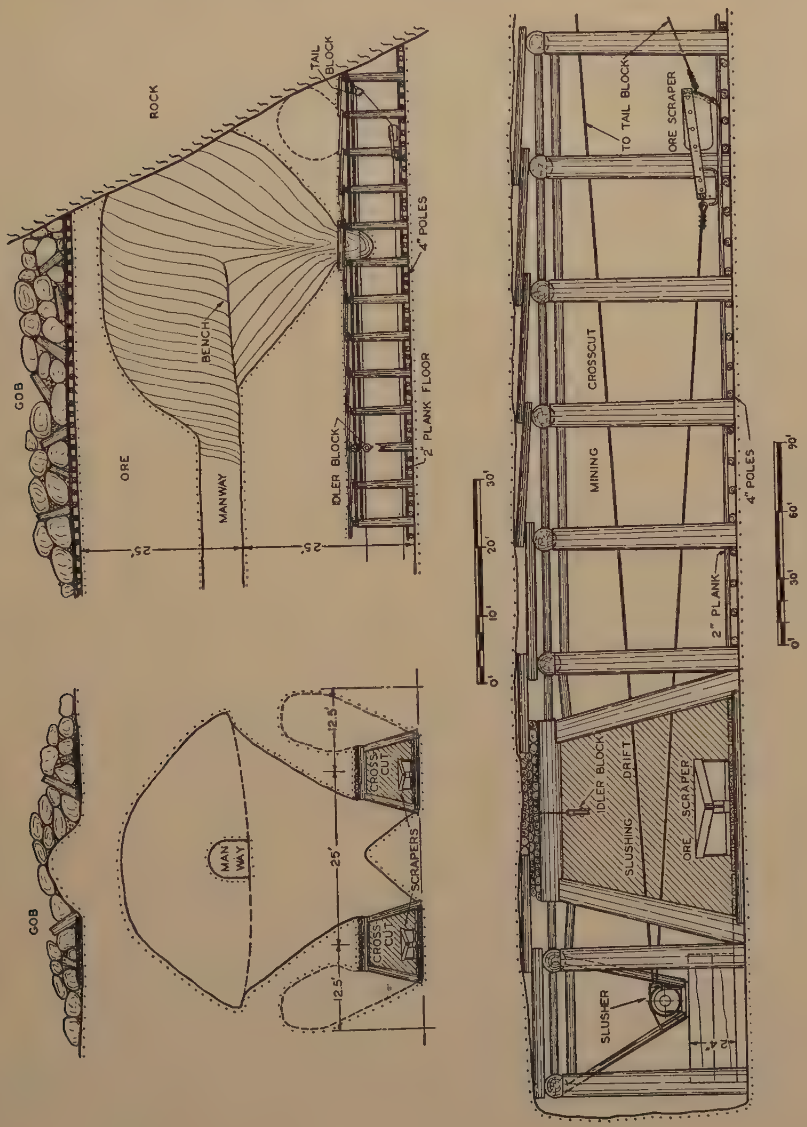


Fig. 4.—Arrangement of stope.
Top left, cross section; top right, longitudinal section.

Timbering

All entry work not gunited is timbered with three-piece sets of 8-ft. timber 11 to 13 in. in diameter. The sets are placed at 5-ft. centers on the main levels and 5½-ft. centers on the sublevels. The caps are framed with a 12-in. horizontal joggle; the posts are cut to fit into the joggle when stood with a rake of 2 in. per foot. The back and sides are lagged with 6-ft. split cedar lagging. All raises are double 4 by 4 ft. inside each compartment, and are closely cribbed with 5½ to 7-in. tamarack framed with 6-in. joggle and when used as ore passes are lined with 2-in. hardwood plank.

All sublevel drifts and crosscuts are poled down crossways with 4 to 5½-in. tamarack poles and covered with 2-in. hardwood planks. The cover makes a good slushing bottom and holds the gob back from entering the stope below through the weak crosscut bottom. Approximately 2 board feet of timber per ton is used in the sublevel caving operations.

Timber is handled from the main-level haulageway directly into the working place when the level pillar is being mined. Timber is received on the sublevels through the timber raise from the level below.

The crosscut between the mining raise and the timber raise furnishes room for storage of timber. There it is loaded on top of the scraper and hauled to the working places.

VENTILATION

Fresh air is distributed to the crosscuts from the quartz-slate drift. It travels through the raises to the mining sublevel, along the slushing drifts to the quartz-slate air raises south of the footwall. High-pressure fans of 5000 to 8000 cu. ft. capacity, with 12-in. metal and fabric tubing, are used to drive the air from the fresh source to the working faces.

The ore is generally moist, contains 3 to 4 per cent free silica, and causes little dust in handling. Filter-pad respirators are worn in ore places, and in rock, which is highly siliceous, air-line respirators only are worn.

SLUSHING

The slusher hoists are 25-hp. continuously rated double-drum planetary drive type with rope speeds of 300 ft. per min. on the pull and 400 ft. per min. on the return.

The scraper, known as the Holcomb, is 22 by 52-in. cast molychrome steel, semi-hoe type, with 6-in. removable lip and 17 by 28-in. side plates, and has a capacity of one ton. The back is curved on a radius of 18 in., the lip is set at a 38° digging angle. All slushing is done on plank except when breasting. In this operation a top plate is used on the scraper, which causes it to ride on its load with less tendency to dig the bottom too deep.

Preformed ropes, 6 by 19 seale construction, with independent wire center, ⅝ and ½ in. respectively, are used on the pull and return ropes.

An 8-in. cast molychrome-steel wheel, bronze-bushed in a cast molychrome frame with a fitting for greasing, is used for tail sheave; a 5-in. pulley of similar construction for idlers.

In the slushing drifts, tail sheaves are hung on the caps. In breasting, they are hooked on cast wedges anchored in the breast. In double slushing the cross-haul hoist is set up 2 ft. above the floor level and the cross ropes work between the ropes of the drift slusher.

All working places have floodlights set beside the hoist. A switch box with red light is operated by a pull cord strung along the sets to signal the slusher operator when anyone desires to enter the slushing drift.

PERFORMANCE

All timber and materials, except explosives, are delivered to the miners in their working places on the main levels and to the storage place on the sublevels.

The task for two men in sublevel work is to advance a breast 5½ ft. in a shift. A wage rate is established for this amount of work and additional footage is paid for at the same rate. The actual average time of each operation is 2 hr.; that is, mucking, timbering and drilling and blasting. Double raises are advanced 4½ ft. per shift by two men.

The task for stoping is 40 tons per man per shift. A bonus is paid for tonnage over this amount. The miners work in gangs of two. When mucking there is one man at each slusher. Two gangs work together in a stope, one gang mucking while the other gang drills and blasts.

A slushing drift with one stope in operation and sufficient crosscutting to keep ahead of the mining produces about 6000 tons per month working double shift five days per week. When the demand has been urgent for a particular grade, 600 tons per day have been produced from one slushing area by increasing the number of miners.

DISCUSSION

(J. Murray Riddell presiding)

W. A. KNOLL,* Ironwood, Mich.—The method described in this paper, mining pillars 50 by 50 ft. in cross section and 50 ft. high, is at least unique on the Gogebic Range, where the average height mined by sublevel mining is approximately 20 ft. The fact that the gob moves slowly and hangs together well makes the mining height possible. The ore apparently is sticky and hard to handle from chutes, therefore the system of scraping it directly into cars is employed, and the extra labor employed is compensated for by the lack of delays in handling sticky ore from chutes. The ore body is developed so that adequate ventilation is supplied to the working face, and it is apparent that a thorough study of underground conditions was made, and a system of mining employed that not only fits the underground conditions but results in a high product per man without any undue hazards involved.

---

* General Superintendent, Mining Department, Pickands Mather & Co.

Shaft Sinking on the Gogebic Iron Range

By W. A. Knoll,* Member A.I.M.E., and J. C. Sullivan†

(New York Meeting, February, 1938)

The sinking of a new shaft at the Newport mine, Ironwood, Mich., was started in May 1931 and completed on Aug. 3, 1932. During this period, 2665 ft. of shaft in granite was completed, at an average advance of 6 ft. per 24 hr. In November of 1930, experimental work was conducted on 65 ft. of shaft through 22 ft. of overburden where ledge was encountered, and this work convinced the operators that all drilling could be done with vertical holes with the blasting relief to a 4½-in. hole drilled vertically in the center of the cut.

The reason for not adopting the conventional V-cut was to get away from damaging the shaft steel. The blasting in a V-cut naturally tends to throw the broken material upward, but with the cut as adopted the relief was in a horizontal direction to a large extent. The scheme of mucking employed made it necessary to keep the steel shaft sets within 50 ft. of the bottom at all times. This could not be done without serious damage with a V-cut when blasting in granite. The cut as drilled also broke 10.83 ft. per round of drilling, which was considerably more than the experience with a V-cut in the same material. The extended cut reduced proportionately the delays for charging, blowing smoke and handling drilling equipment in the shaft.

Another advantage was the ease in drilling vertical over inclined holes, and the preservation of the proper spacing of the drill holes at the bottom of the cut.

Drilling

Air was carried down the shaft in an 8-in. flanged pipe line at a pressure ranging from 95 to 100 lb. This line was extended to within 100 ft. of the bottom, where a valve and a 4 by 4 by 6-ft. tee was connected to it. To each side of the tee was connected a nipple and elbow, the resulting span of the two elbows being about 4 ft. Lengths of short 3-in. hoses

---

Manuscript received at the office of the Institute Nov. 26, 1937. Issued as T.P. 887 in Mining Technology, May 1938.

* General Superintendent, Pickands Mather & Co., Ironwood, Mich.

† Mining Engineer, Pickands Mather & Co.

(Fig. 1) were then connected to the elbow to bring the line to the last set of steel. At this point the 3-in. hose of the header (Fig. 2) was connected and the header adjusted to the desired height above the bottom by means of a 1-ton chain block suspended on the last set of steel. The 25-ft. length of 3-in. hose was connected to a 4-in. elbow. To this elbow a 4 by 6-in. nipple, fitted with a clamp to which a clevis could be attached, was connected, and another 4-in. elbow with a short 4-in. pipe 7 ft. 3 in. long brought the line to the header proper.

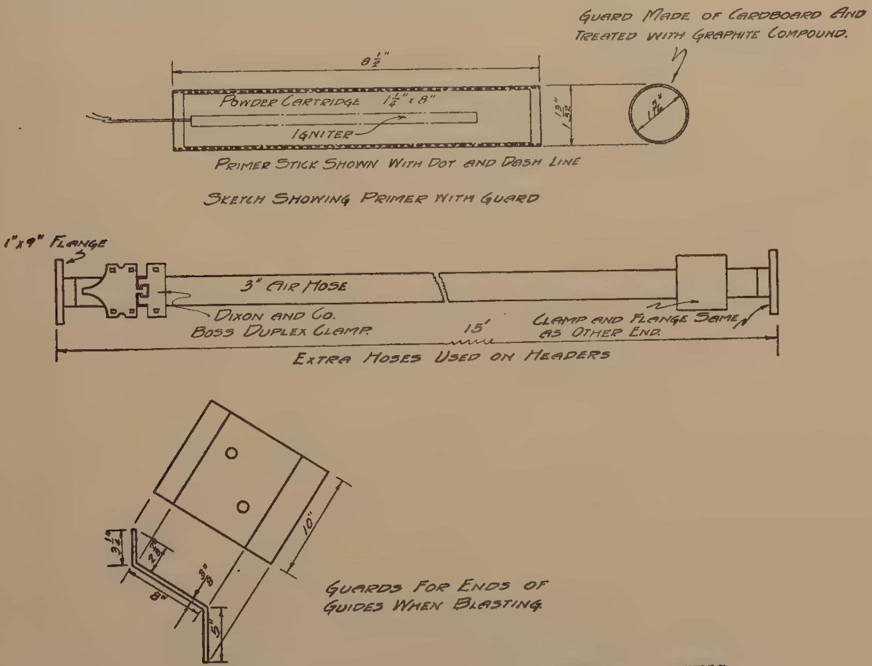

Guard Made of Cardboard And Treated with Graphite Compound.
8½"
Powder Cartridge 1¼" x 8"
Igniter
1⅝"
1 5/32"
Primer Stick Shown With Dot and Dash Line
Sketch Showing Primer With Guard
1"x9" Flange
3" Air Hose
Dixon and Co. Boss Duplex Clamp
15'
Clamp and Flange Same as Other End
Extra Hoses Used on Headers
10"
3½"
2⅜"
3/8"
8"
5"
Guards For Ends of Guides When Blasting

FIG. 1.—GUARDS FOR ENDS OF GUIDES WHEN BLASTING.

The header itself was made of a casting 3 by 9 in., which was bored out in the center. Nine evenly spaced holes were then drilled in the side and tapped for 1-in. nipples. To the nipple was attached a No. 1571 Lukenheimer valve with two additional nipples and an elbow, which reverted the line to a vertical position. An Ingersoll-Rand type E line oiler was connected and to this a 1-in. hose and drilling machine. Each header supplied connections for seven machines and two blowpipes; 12 or 13 machines were used when drilling. The arrangement of the blowpipe connections differed from those of the machine connections in the position of the valve and the omission of the oiler. The velve was placed in the hose line so that the miners might operate it easily while blowing holes. Two headers were used in drilling, allowing 14 machines and four blowpipes to be in operation. The headers were transported up and down the shaft by means of wire-rope slings underneath the cage, and when not in use they were hung at the collar. They were taken down by two members of the crew detailed to that work, so that drilling might be started as soon as the mucking was finished.

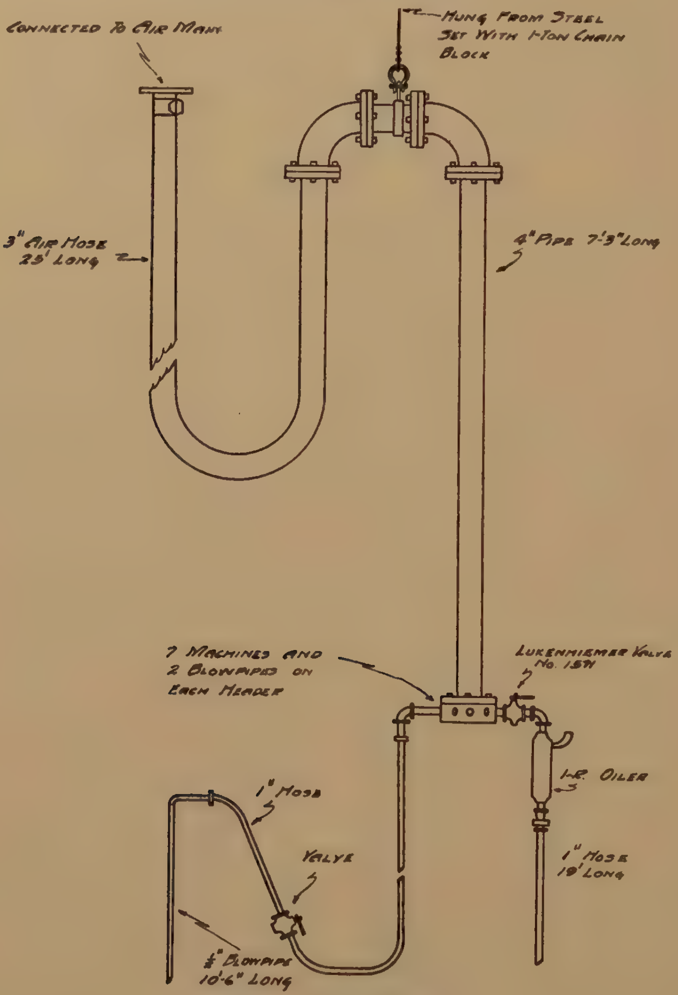

Connected to Air Main
Hung From Steel Set With 1-Ton Chain Block
3" Air Hose 25' Long
4" Pipe 7'-3" Long
7 Machines and 2 Blowpipes on Each Header
Lukenhiemer Valve No. 15H
I-R Oiler
1" Hose
Valve
1" Hose 19' Long
1/2" Blowpipe 10'-6" Long

FIG. 2.—AIR HEADER.

The drilling machines selected for all except the center hole were 3-in. dia. sinkers, weighing approximately 72 lb. Weights were designed to hang on the machines (Figs. 3 and 4). These weights proved to be a very noticeable help in the drilling operation. One-inch hollow hexagon steel was used and starters of both four and six points were employed. The bits of all but the starter were of the four-point cross type and were made up in lengths to 13 ft. with 12-in. runs. The diameter of the starter

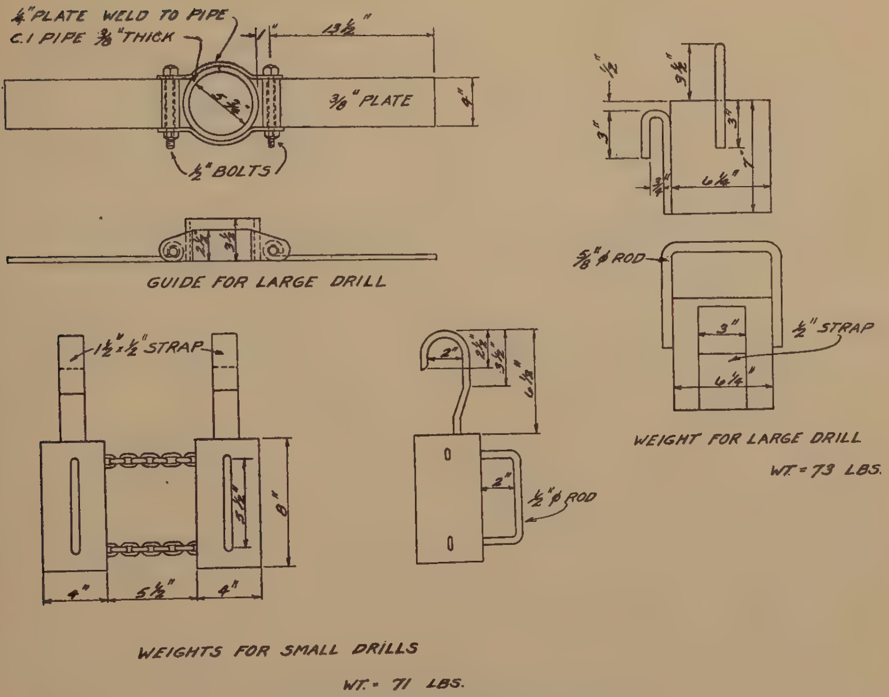


FIG. 3.—DRILL WEIGHTS AND GUIDE.

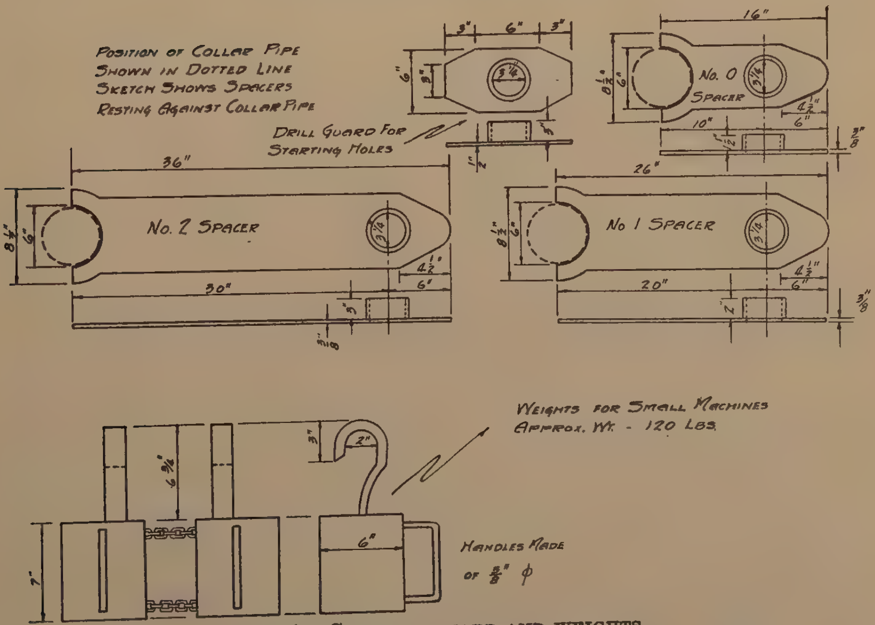


FIG. 4.—SPACERS, GUARD AND WEIGHTS.

was 2⅞ in. and, with a ⅛-in. change for each succeeding length, the diameter of the 13-ft. length was 1½ in. Beyond this diameter the hole was too small for proper loading. Spacers designed to start these holes saved much time (Figs. 3 and 4). Fig. 4 illustrates the spacers used to drill the No. 0, No. 1 and No. 2 holes, with the big-hole collar pipe used as the base for measurements.

Tools for recovering broken bits and for removing bits that were held fast in the hole are shown in Fig. 5. The drill pullers (Fig. 6) were designed to utilize the drilling machine in an inverted position, and, with exceptionally tight bits, to employ the use of two machines. The machine selected for drilling the large center hole was a 4-in. dia. drifter equipped with handles. Weights were designed to hang on the machine (Fig. 3). The use of weights resulted in better drilling speeds with less vibration of the machine. The drill steel was 1½-in. hollow round having six-point starters of 5⅜-in. dia. and four-point starters with 5¼-in. dia. Changes of ⅛ in. were made for succeeding lengths and the longest drill 13 ft. 6 in. long with a bit 3⅞-in. dia. Fig. 3 shows the guide for starting the large center hole with ⅜- by 13½-in. plate used for standing on and holding the drill firm.

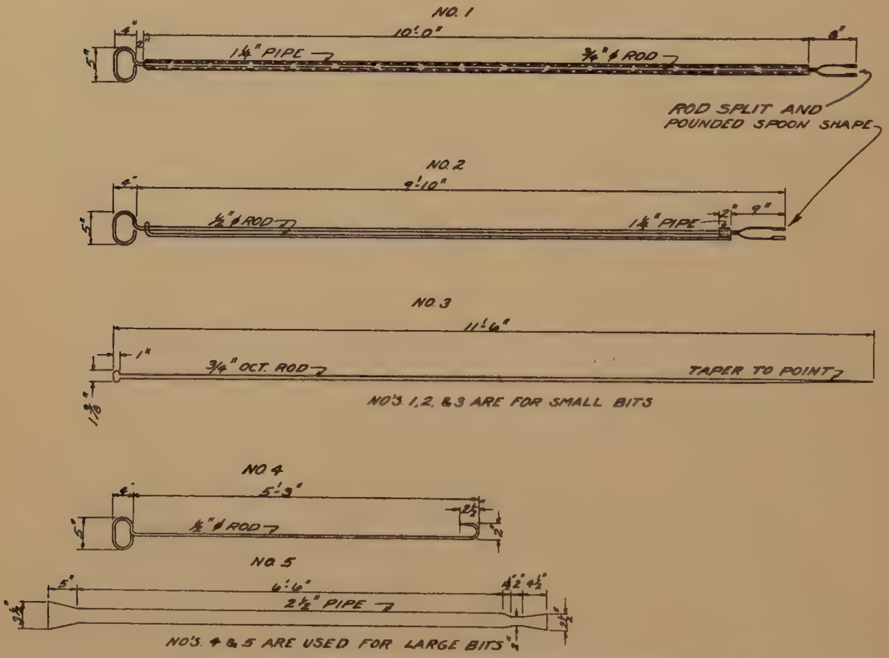


FIG. 5.—TOOLS FOR RECOVERING BROKEN BITS.

Several plans were followed in the drilling, depending on the character of the ground, but Fig. 7, with 68 holes, was the plan that gave the best

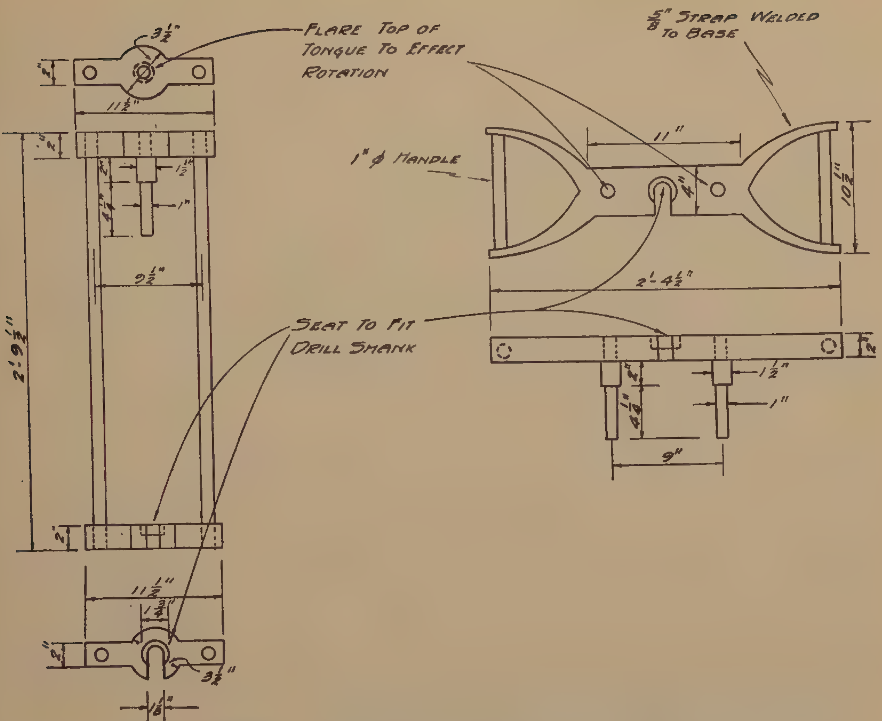
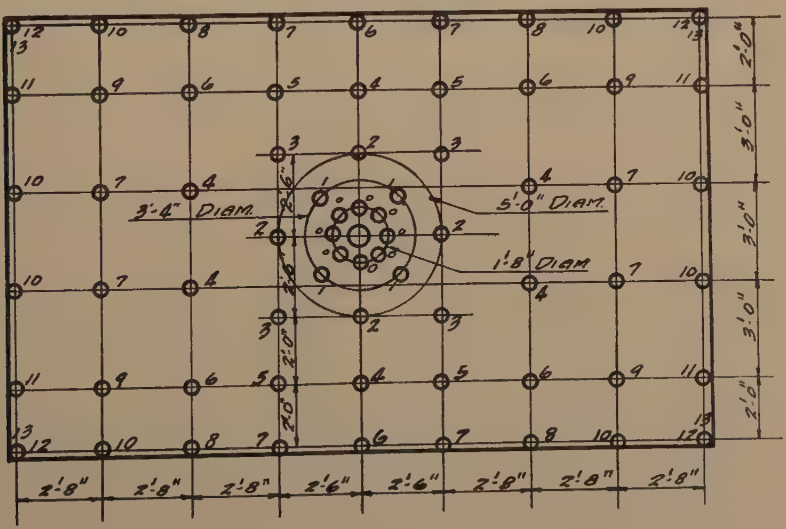


Fig. 6.—Drill pullers. Rods and tongue made of drill steel, bases of machine steel.



In The Corner Holes
A No. 12 Delay is Placed
At Top And No. 13 Delay
At Bottom.

Fig. 7.—Blasting plan used for new shaft, Newport mine.

results. Plumb lines were dropped from the corners of the steel and one plumb line dropped from the center for the large hole. The miners then stationed themselves in position to drill with as little interference from each other as possible, three men being detailed on the center hole. The large hole was drilled slightly deeper than the other holes, its purpose being a recess for the other holes to break to. The time required to drill the large hole was about 2 hr., and upon its completion the same three men began drilling the No. 0, No. 1 and No. 2 holes. The No. 0 and No. 1 holes were inclined slightly toward the center to make up for the decreased bottom diameter of the large hole. All other holes were drilled vertically. After a hole was started, a collar pipe made of 3-in. pipe, 12 in. long, was inserted into the top of the hole to keep choppings and sludge from going back into the hole. Upon the completion of the hole, the collar pipe was replaced by a cedar plug, which was left in place until the hole was ready to be charged.

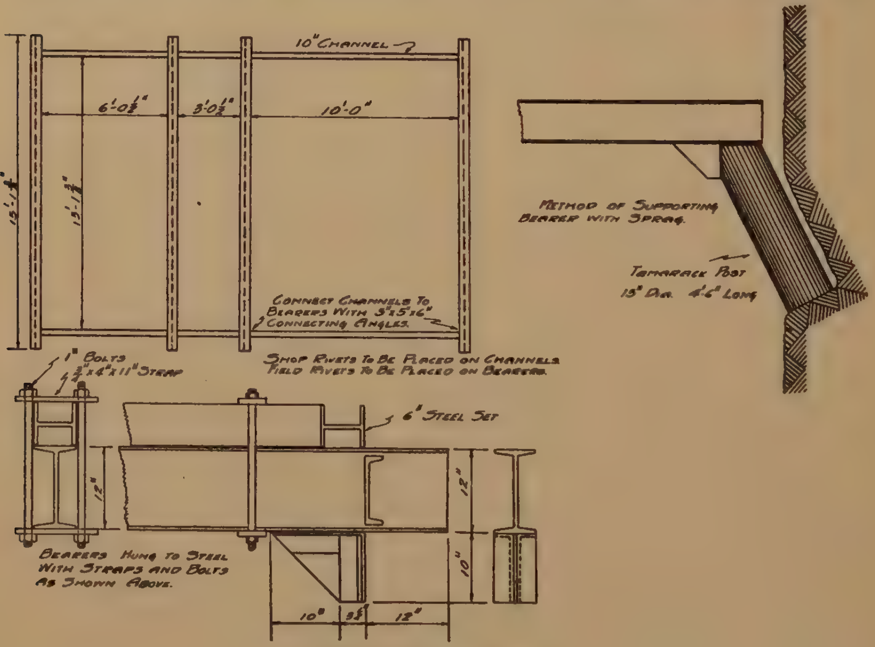

10" CHANNEL
6'-0½"
3'-0½"
10'-0"
15'-1½"
13'-1¾"
CONNECT CHANNELS TO BEARERS WITH 3"×5"×6" CONNECTING ANGLES.
1" BOLTS
¾"×4"×11" STRAP
SHOP RIVETS TO BE PLACED ON CHANNELS
FIELD RIVETS TO BE PLACED ON BEARERS.
METHOD OF SUPPORTING BEARER WITH SPRAG.
TAMARACK POST
15" DIA. 4'-6" LONG
6" STEEL SET
12"
12"
10"
BEARERS HUNG TO STEEL WITH STRAPS AND BOLTS AS SHOWN ABOVE.
10"
3½
12"

FIG. 8.—BEARING SET.

Sharp steel was supplied and dull steel removed at short intervals during the drilling shift to avoid any unnecessary congestion that might result from an accumulation of steel. This was done with the cage and did not interfere with the drilling because the cage was stopped about 8 ft. from the bottom and steel handled from that point. Approximately 800 pieces of steel were used with each drilling round, the breakage ranging between 50 and 100 pieces. The average depth of holes drilled was

12½ ft. with bottoms of 1½-in. dia. The lineal footage drilled, excluding the large hole, amounted to approximately 850 ft. per round, using the 68-hole plan. The average drilling time for 247 cuts taken in the shaft was 9½ hours.

Hitches for bearer sprags (Fig. 8) were drilled in the regular drilling shift. Six holes, 3 ft. 6 in. deep, were drilled on 9-in. centers with the two lower holes horizontal, the two center holes inclined up at about 30° and the two top holes inclined up at about 45°. They were fired with delays, the top holes breaking first, the middle two second and the lower two breaking last. Eight groups were drilled and blasted whenever the installation of bearers made it necessary.

Water presented no difficulty, there being not much more than enough required for drilling. The flow during normal weather amounted to about 4 gal. per minute and excess water was disposed of by pumping with a portable centrifugal pump, with a 3-in. discharge, into the skip, which was suspended about 15 ft. from the bottom. When the skip was filled, it was hoisted to surface and the water dumped through the pocket into a launder, which carried it away from the shaft.

CHARGING

Powder used for blasting was 80 per cent gelatin in cartridges, 1¼ by 8 in. No. 8 electric blasting caps and delay igniters were employed with the order of firing shown on Fig. 7. The primers were made up in the powder house, all tagged with their proper number and given an application of waterproofing compound by the powder man. They were then inserted into the powder stick and the outside paper covering was tied tightly again. All igniters were tested with a galvanometer before being used, to ensure no misfires from that source. After the primer was ready, it was inserted into a strong cardboard guard (Fig. 1), to which an application of glue was given on the inside. This guard was used to ensure against any premature blast by sharp contact in rough holes when charging. The ends of the igniter conducting wires were then cleaned with sandpaper to make sure of good connections. As they were finished, the primers were divided and placed in two wooden boxes, one to be used in each half of the shaft. Sticks of tamping were made of moist clay in cartridges, 6 in. long. Powder was placed in canvas bags, and when needed primers, tamping and powder were sent to the bottom.

The blasting switch on surface was a double-throw, double-pole switch, one throw short-circuiting the blasting circuit. The switch box was kept locked and the shaft foreman had the only key. The power line was carried down the shaft to within 200 ft. of the bottom, where it too was short-circuited. It was wound on a reel and could be pulled out to any length needed. The ends of the power line were taken to the last set and twisted together until the final connecting was done. The power cable used was No. 8, a 49-strand two-conductor rubber-covered cable.

The holes having been blown out after drilling, and all mechanical equipment removed to the surface, the electric lights in the shaft were turned off at surface and the process of charging was begun under the supervision of the crew foreman. Four sticks of powder were placed in the bottom of each hole and followed by the primer; then the remainder of the powder allotted to each hole, with the necessary tamping on top. Tamping was done with a 1¼-in. round wooden tamping stick. The number of sticks of powder varied for different holes. The No. 0 holes were charged with eight sticks of powder and six sticks of tamping were used. In holes Nos. 1 to 11 inclusive, 14 sticks of powder were used with 3 sticks of tamping. In the No. 12 corner holes two delays were used (Fig. 7). These were charged by placing two sticks of powder in the bottom; then the No. 13 delay; then 11 sticks of powder with clay tamping on top to within 4 ft. of the collar; then one stick of powder on top of the tamping followed by a No. 12 delay; then four sticks of powder, and finally tamping tight to the collar of the hole. This method of double delays brought out the corners in a very satisfactory manner. The ends of the connecting wires were then collected in groups and a double No. 20 lead wire was brought down from the last shaft set, and, circling the shaft, was connected directly to the instantaneous No. 0 holes. Connecting in parallel, one side of each group was connected to the lead wire and taped. When all connections on the one wire were made, the second double No. 20 lead wire was brought down and the other connections were made as for the first wire. All tamping sticks, bags, boxes, etc., were then loaded on the cage and the crew was hoisted to the bottom set. Here the first connection was made by the foreman, who connected the lead wires to the power line that had been dropped to the last set. The connections were made firm and taped and the cage hoisted to the power-reel elevation. Here the second connection was made and the crew hoisted to the surface, where the blasting switch was unlocked and the charge fired by the throwing of the switch. After the charge had been exploded, the switch was again short-circuited and the blasting box locked until needed. The power used was 250-volt, direct-current, assuring about one ampere per igniter.

VENTILATION

Ventilation in the shaft was maintained by the use of a blower driven by a 15-h.p. motor. The fan was connected to a 12-in. 18-gauge steel pipe, which terminated about 100 ft. above the bottom. To the end of the steel pipe was attached 75 ft. of Ventube, which was pulled up to the steel pipe just before the blast. This was accomplished by means of a small winch at the collar, with ½-in. hemp rope. Immediately following the blast, it was lowered again by the same method and about 30 min. was needed to clear the bottom of smoke and gas. The fan is capable of delivering 3600 cu. ft. per min. at surface and 1500 cu. ft. at a depth of 3000 feet.

MUCKING

After an elapse of 30 min., the smoke and gas had been diluted sufficiently to allow men to return to work. The first step was to remove all loose rock that might have been thrown up and lodged in precarious positions on the steel, and this was accomplished by the crew going slowly through the shaft, using the cage and skip. The side walls below the steel sets were then carefully inspected for loose rock resulting from the blast and all loose rock was barred down. This accomplished, the setting up of mucking equipment began. The bucket method of mucking was discarded in favor of mucking trays (Fig. 9). The tray is 5 ft. long, 30 in. high and 42 in. wide, with the dumping end and top open. The tray rested on top of the muck pile and the broken rock was shoveled into it. Two trays were used, each having a capacity of about 35 cu. ft. They were made of ⅜-in. plate with all joints welded together. A bail was placed as shown, so that the center of gravity was just back of the shaft bolts of the bail, thereby preventing the tray from tipping forward when raised to dump. The raising and dumping of the trays was accomplished

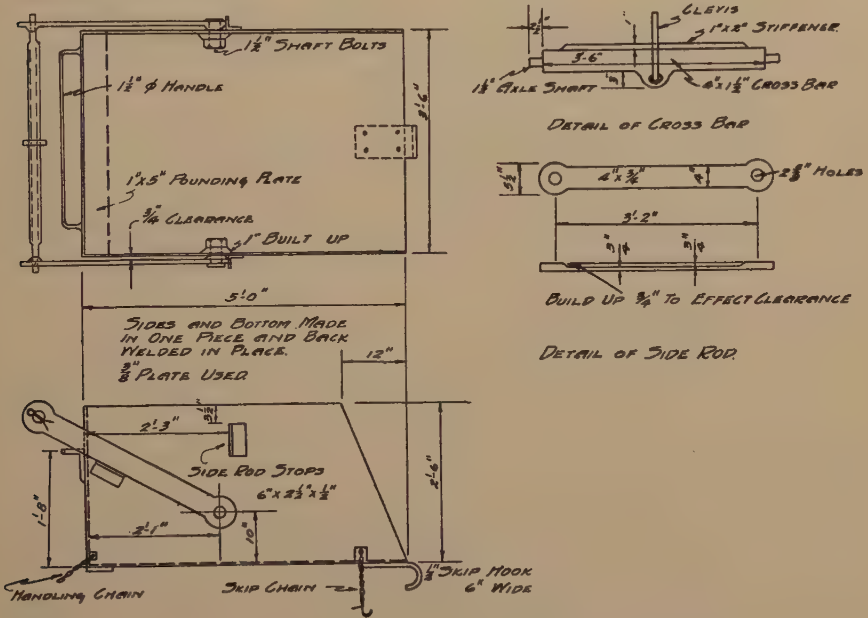
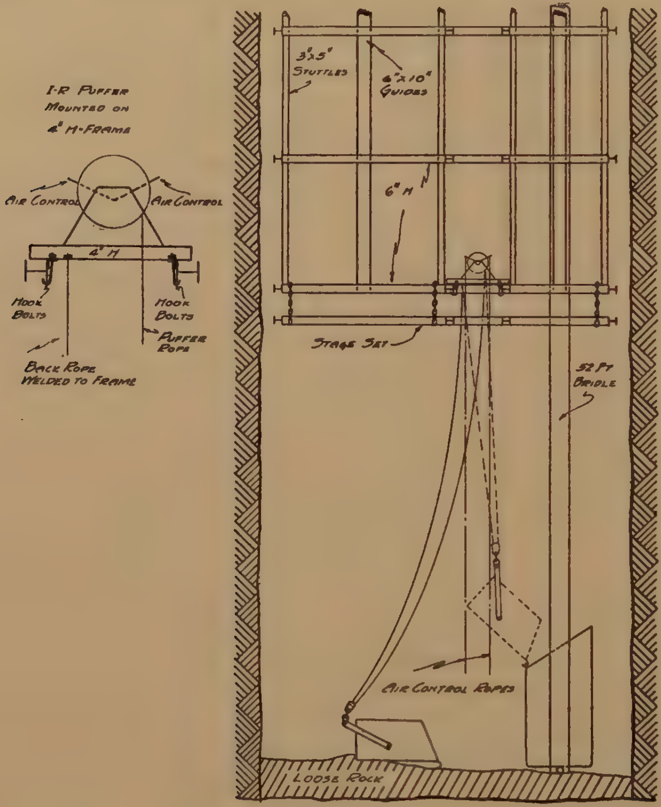


FIG. 9.—MUCKING TRAY.

by two 10-hp. single-drum hoists driven by compressed air (Fig. 10). The hoists were mounted on 4-in. H-frames, and as units were set in place on the steel in the pipe and ladder compartments and made fast to the steel by use of hook bolts. When the mucking period was completed, trays and hoists were taken to the surface, in the cage, and inspected and oiled for the next mucking cycle. Air was furnished to the hoists by means of tapped connections to the air line and carried from that point to the hoists by 1-in. hoses. Ropes suspended as shown in the figure enabled the hoists to be controlled from the bottom of the shaft. During the filling of the tray, the skip, having a capacity of approximately 75 cu. ft., was stationed at the safety mark. When needed, it was rung to the bottom and the tray raised and dumped. A hook on the tray slipped over a bar on the skip and by further raising the back end of the tray dumping was effected. (Dumping of the trays was also accomplished by means of skip chains with hooks attached, as shown in



Full Lines Show Tray in Loading Position
Dotted " " " Dumping "

FIG. 10.—OUTLINE OF OPERATION OF TRAYS.

Fig. 9.) The tray was then lowered and by means of a chain was pulled to any desired position on the bottom. The skip was then raised to a height of about 15 ft. above the bottom to wait for its second tray, which was dumped as soon as it was filled, after which the skip was sent to surface. By interchanging the rope blocks, both trays could be dumped into the same skip, and enabled mucking to proceed without interruption.

The side walls of the shaft were carefully watched for loose material, being trimmed continuously as the mucking process continued. When mucking progressed to the point where the shattered bottom was encountered, one drilling header was lowered and connected up. Four 1⅝-in. dia. "pavement breakers" with spades and diggers, for ripping up the tight material in the bottom, were connected and the mucking was continued with one tray only. At this period in the mucking cycle, the work progressed at a slower rate, and one tray served the operation adequately while allowing more space in the bottom in which to work. The mucking was continued in this manner until a firm, solid bottom was reached. The amount of broken rock removed during the mucking cycle averaged about 250 cu. yd. Excess water was pumped into the skips with the rock and hoisted to surface.

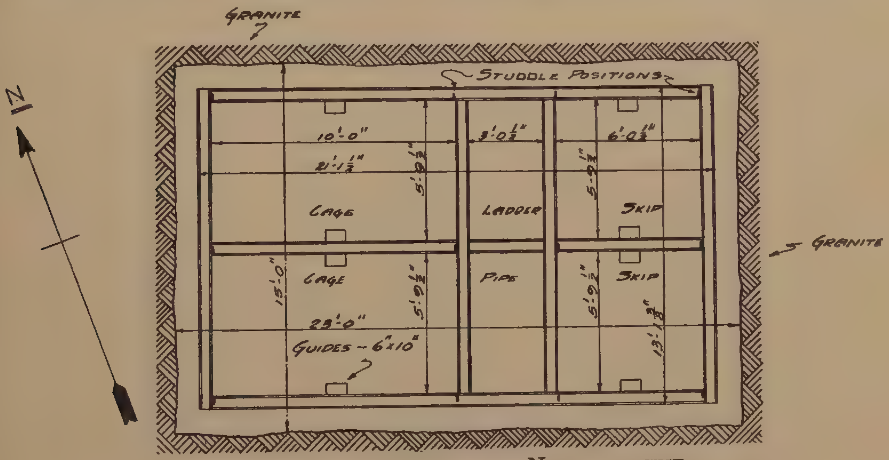


FIG. 11.—PLAN OF NEW SHAFT, NEWPORT MINE.

Thirty minutes before the expected completion of mucking, two men were sent for the second header and other drilling equipment. This equipment was then put in readiness for use and the mucking equipment loaded on the cage, enabling the drilling operations to begin without delay. The tray was swung into the cage, and all shovels, picks, machines and other equipment loaded with it. The cage then ascended to a hoist position, where the hoists were taken aboard and removed to surface. The operation of unloading the cage on surface was facilitated by the use of a jib crane.

Mucking was experimented with, using one large tray instead of two smaller ones. It proved too large to be handled easily and was discontinued in favor of the two small ones. A scraper was also tried out in the mucking operation, but was not efficient because of the very confined space in which it could travel and accumulate its load. The average mucking time for 247 cuts was about 23 hours.

SHAFT CONSTRUCTION

The essential features of the shaft set are shown in Fig. 11. The steel set is made of 6-in. H-sections (23 lb.). The sets are hung on 8-ft. centers with studdles made of 3 by 5-in. angle iron, and a 12-in. bearing set (Fig. 8) was installed at intervals of about 150 ft. Steel was installed after each blast unless the distance between the last set and the bottom was less than 35 ft., as considerable damage to steel from blasting resulted when the steel was carried too close. The maximum distance was governed by the 50-ft. bridles on the skips and cages (Figs. 12 and 13). Consequently, the distance from the steel to the bottom ranged between 35 and 50 ft. The time of shaft-steel installation was deferred, if possible, to the morning shift, but if necessary was installed immediately after the blast.

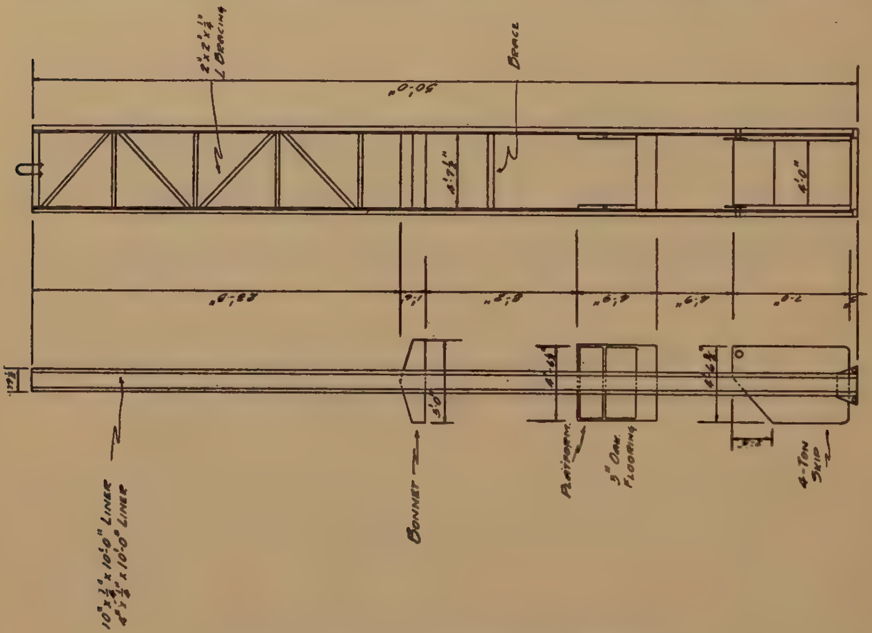


FIG. 12.—SKIP AND BRIDLE ARRANGEMENT.

A stage or dummy set was used, hung by chains 2 ft. below the last regular set of steel (Fig. 10). The set was a duplicate of the shaft set with tamarack poles tied to the underside between the flanges of the members. The stage set took the brunt of the blow in blasting, and very few replacements were made of the shaft steel during the entire sinking.

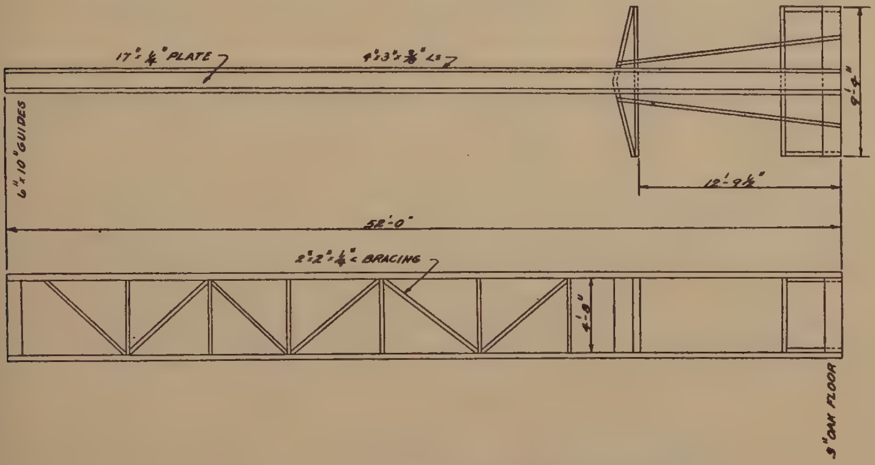


FIG. 13.—CAGE AND BRIDLE ARRANGEMENT.

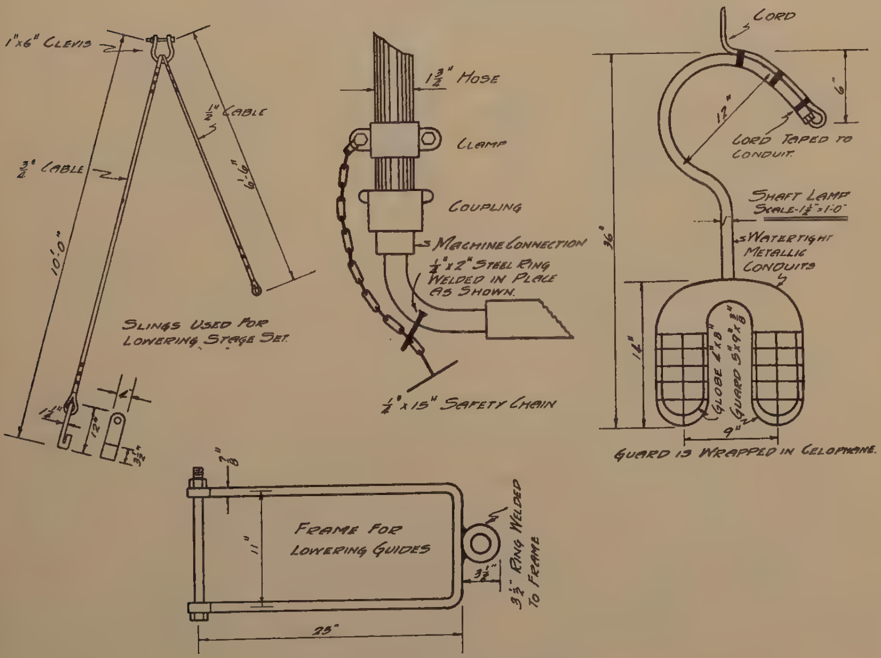
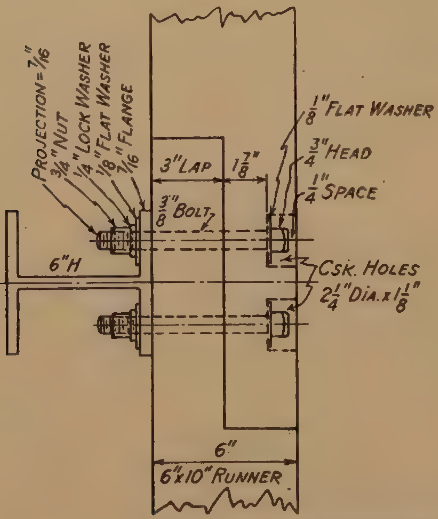


FIG. 14.—SHAFT EQUIPMENT.

The stage set had to be replaced whenever the members got so badly bent and twisted as to render them incapable of serving their purpose. When removal of the stage set became necessary, it was cut with a torch and allowed to fall to the bottom, where the pieces were loaded on the cage and removed to surface.

The first step in the installation of steel was the lowering of the stage set 10 ft. below the last set in place. This was done by means of four slings (Fig. 14) in the following manner:

Four 1-ton chain blocks were hooked on the set above the bottom set and the ½-in. cable was hooked through the thimble. The clevis shown was bolted to the stage set through a hole drilled in the inside flange on the wall plates. The stage was raised to detach the chains and then lowered until the weight was taken by the ¾-in. cable as the 4 by 12 by ¾-in. hooks were hung on the last set. All four corners of the stage were lowered simultaneously, making the distance below the set 10 ft. Next, the stage planks were placed and the installation of the wall plates started. All steel was carried down the shaft in the cages or suspended under them. After the two wall plates were hung on studdles, the end plates and dividers were installed and the set was lined and blocked. Two lining wires with a 1-lb. weight were carried down diagonal corners from the plumbed set above. Sprags were cut on surface in lengths varying from 8 to 20 in. and only an occasional piece had to be cut by timbermen. The sprags were placed at both ends of dividers, wall plates and end plates and wedged tight with pine wedges. Lathes were made of 2-in. hardwood plank, 6 and 8 in. wide, cut 7 ft. 11 in. long to facilitate its installation. The lath was made secure by the use of 2¾-in. filler pieces wedged behind the inside flange of the steel. Two sets of steel were installed and lathed during one installation period.



1 Projection	7/16"
1 Nut	3/4"
1 Lock Washer	1/4"
1 Flat Washer	1/8"
1 H-Section Flange	7/16"
1 Lap	3"
1 Lap	1 7/8"
1 Flat Washer	1/8"
Total Length of 3/8" Bolt	7"

16 Bolts per Set
342 Sets @ 16 Bolts = 5,472 Bolts

Fig. 15.—Bolts for runners for new shaft.

The guides of 6 by 10-in. dressed yellow pine, 85 per cent hearts, were framed to span two sets. They were fastened to the steel with ¾ by 7-in. bolts and framed so that the lap came at the set (Fig. 15). The guides for the skip roads were lowered in the skip to within a short distance of the bottom. A ¾-in. single and double block, hung on the steel above, was then attached to the guide and the skip was lowered, after which the guides were lowered and bolted into place between the skip and the steel. The guides for the cage roads were lowered by suspending them below the cage and then swung into place with a rope and block. A clevis was used in lowering the guides (Fig. 14). A hole was drilled through the guide about 18 in. from the end and the bolt shown in the frame was passed through the guide, facilitating its handling. Guards were placed at the bottom of the guides to prevent damage to them from blasting (Fig. 1).

After the installation of the sets, a length of pipe for the air line was lowered and installed. A ¾-in. strap, with a ring for the hooking of a clevis, was bolted about the pipe and the pipe lowered under the cage. At the point of installation, it was slid across the divider and into the pipe road where it was handled with a chain block until bolted in place.

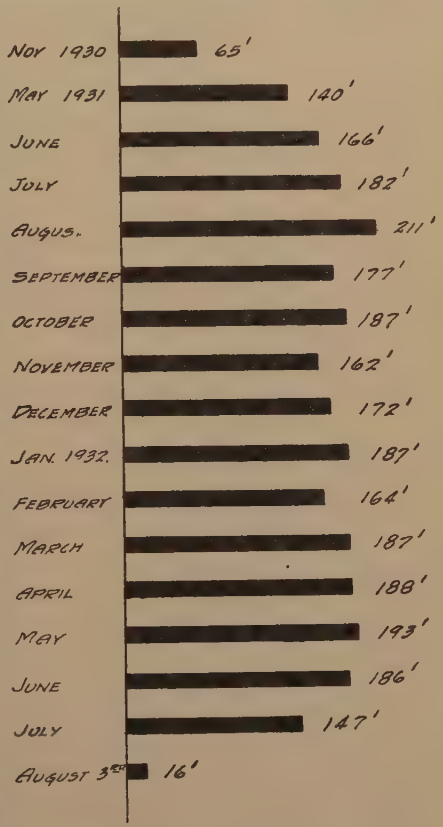


Fig. 16.—Advance in work on new shaft, Newport mine.

Bearing sets were installed at intervals of about 150 ft., the bearers being of 12-in. I-beams, spaced by a 10-in. channel iron (Fig. 8). In the installation of the bearing set, the bearers were strapped to the shaft set with ¾-in. straps, as shown in the figure. The channels were then set in place and 15-in. tamarack sprags supported the bearers. These bearers were concreted in after the completion of the shaft sinking. To ensure the bearing set being level, it was customary to level up a shaft set a short distance above the elevation at which the bearers were to be installed.

Sollars made of electroforged grating were installed, made in two pieces and fastened to the steel by means of hook bolts.

Electric Wiring

All bell, power and light lines were carried down the shaft in the ladder road. At the bottom of the shaft a lamp, shown in Fig. 14, was suspended about 15 ft. above the bottom. A covering of cellophane over the globe guard protected the globes from breaking when drill holes were being blown out, and at the same time did not materially decrease the intensity of the light. A second lamp was hung on the shaft steel at the safety mark.

Sigaphones were in use on both cages, enabling the cages to be stopped or started at any point in the shaft. A telephone enabled conversation to be carried on from the cage to the collar of the shaft and powerhouse. A system of bell signals was used also, to indicate to the top landers what material was needed at the bottom.

Labor

Work progressed in the shaft on four 6-hr. shifts, working seven days per week. The crew was organized as follows:

1 captain
4 shift bosses
52 miners
2 construction men
1 electrician
1 pipeman
3 landers
3 truck drivers
6 hoisting engineers
5 shopmen
1 powderman
—
Total..79 men

All labor except bosses and miners worked on an 8-hr. schedule and shop labor was performed on one 10-hr. shift only.

In all, 444 days were worked in the sinking of 2665 ft. An average of 6 ft. per day was maintained, with an average of 178 ft. per month. The maximum footage made was 211 ft. per month while the minimum was 140 ft. (Fig. 16).

Miscellaneous

When sinking had progressed to the level elevation, it was suspended temporarily and two crosscuts 6 by 6 ft. were driven on each side of the shaft. This work was done using the 3-in. dia. sinking machines and shoveling the broken rock into the skip. Six-foot holes were drilled and 24 ft. of crosscut was driven on each side, to facilitate the cutting of the shaft station later on.

An opening 6 by 6 ft. was also carried down on the pocket side, to be boarded up and used as a raise when excavating for the loading pocket.

DISCUSSION

(*J. Murray Riddell presiding*)

Member.—How many pounds of powder were used per cubic yard? What was the effect on the walls of the shaft as left? Was there not considerably more shattering effect than there would have been with the V-cut?

J. M. Riddell,* Duluth, Minn.—My impression was that the walls of the shaft as nearly as could be were very well preserved. There were a great number of delayed shots there, so the relative burden in any one particular case was not too great.

Member.—Was the rock jointed noticeably?

J. M. Riddell.—In places.

V. D. Johnston,† Cleveland, Ohio.—Generally it is pretty uniform rock.

Member.—Did slabs follow the joints back on the sides after the shot?

J. M. Riddell.—That would be the exception.

G. S. Rice,‡ Alexandria, Va.—Does any subsequent lapping have to be done on the shaft or anything of that kind for taking care of the rock that might have been spoiled?

V. D. Johnston.—I believe the shaft is now lapped throughout; some of the shafts are gunited. I believe that in this instance the shaft was lapped and then, perhaps, gunited.

J. L. G. Weysser,§ Lansford, Pa.—Not as part of the discussion, but as a matter of personal curiosity, I am prompted to ask if the burnt-cut method has been used where there is a pronounced joint system, as through sediments for instance, rather than through a granite, and was it successful?

F. J. Smith,‖ Ramsay, Mich.—I have not tried the burnt-cut method in shaft sinking in slates, but did try it in drifting in ferruginous slates. Apparently it has been used successfully in driving drifts in granite but it did not work out satisfactorily in the slate formation on which we tried it.

V. D. Johnston.—It is being tried out at Newport in drifting in granite, apparently successfully.

T. T. Read,¶ New York, N. Y.—Why was the steel breakage so high?

R. L. Healy,** Kirkland Lake, Ont.—How many machines were drilling out?

W. A. Knoll (author's reply).—About 3 lb. of powder was used per cubic yard in the shaft sinking. This compares with about 2¼ lb. per yard in the same material —that is, granite—using a V-cut.

We believe that probably the use of weights on the drills added considerably to the steel breakage, although the drilling speed was increased and the vibration of the machine was considerably dampened thereby. Twelve machines were drilling at one time.

---

* Assistant Manager, Republic Steel Corporation.
† Oglebay Norton & Co.
‡ Consulting Mining Engineer.
§ Lehigh Navigation Coal Co.
‖ General Superintendent, Castile Mining Co.
¶ Professor of Mining Engineering, School of Mines, Columbia University.
** Wright-Hargreaves Mines.

Sand Filling at the Homestake Mine

By A. J. M. Ross,* Member A.I.M.E.

(New York Meeting, February 1939)

Backfilling of stopes and other underground openings in the Homestake mine with sand tailings was undertaken primarily to reduce surface subsidence, which was wrecking much of the surface plant and a substantial part of the town, and to make possible maintenance of openings into the top of pillar stopes as mining progressed. The progress of this type of filling has been rapid and the results very satisfactory. Sand is now used in filling timbered stopes, shrinkage stopes, and abandoned drifts and crosscuts. A permanent pipe-line system for transporting sand tailings into the mine by gravity has been installed. The close filling of all stopes has resulted in important benefits other than the two major ones.

The ore bodies in the Homestake mine occur as replacements in one distinctive layer of the metamorphosed pre-Cambrian sediments along a group of minor folds that together form a complex anticline that pitches at 35° to 40° to the southeast. The ore bodies are extraordinary for their magnitude. Widths of over 300 ft. have been mined and continuous ore has been followed for over 1200 ft. along the strike on some of the upper levels.

Mining has been by shrinkage stoping with square-set mining of crowns and pillars. In the standard plan, shrinkage stopes 60 ft. in the direction of strike and the full width of the ore have been taken up to within 25 ft. of the level above. After filling, the crown has been mined by square-set methods, as have also the 40-ft. pillars between shrinkage stopes. In former days, when the supply of ore seemed inexhaustible, the mining of pillars was allowed to lag far behind because of the high cost. Many pillars were rather badly fractured on the level and in some places between levels.

Coarse waste was used for backfilling all stopes, but in pillars it became increasingly difficult to maintain haulageways for the transportation of this fill over the top of the stopes as mining progressed. The cost of maintaining these crosscuts was excessive and in many pillars practically impossible. Therefore it became the practice to fill pillar stopes by tapping the back and sides of the stopes.

Manuscript received at the office of the Institute Jan. 27, 1939. Issued as T.P. 1075 in Mining Technology, July 1939.

* Mine Superintendent, Homestake Mining Co., Lead, South Dakota.

With coarse waste it is impossible to secure a tight fill. The percentage of fill depends on the relative amounts of coarse and fines in the filling material. With the character of fill available it is estimated that coarse waste gave approximately 50 per cent fill. With tapping of fill in pillar stopes, the total volume mined was approximately 30 per cent filled.

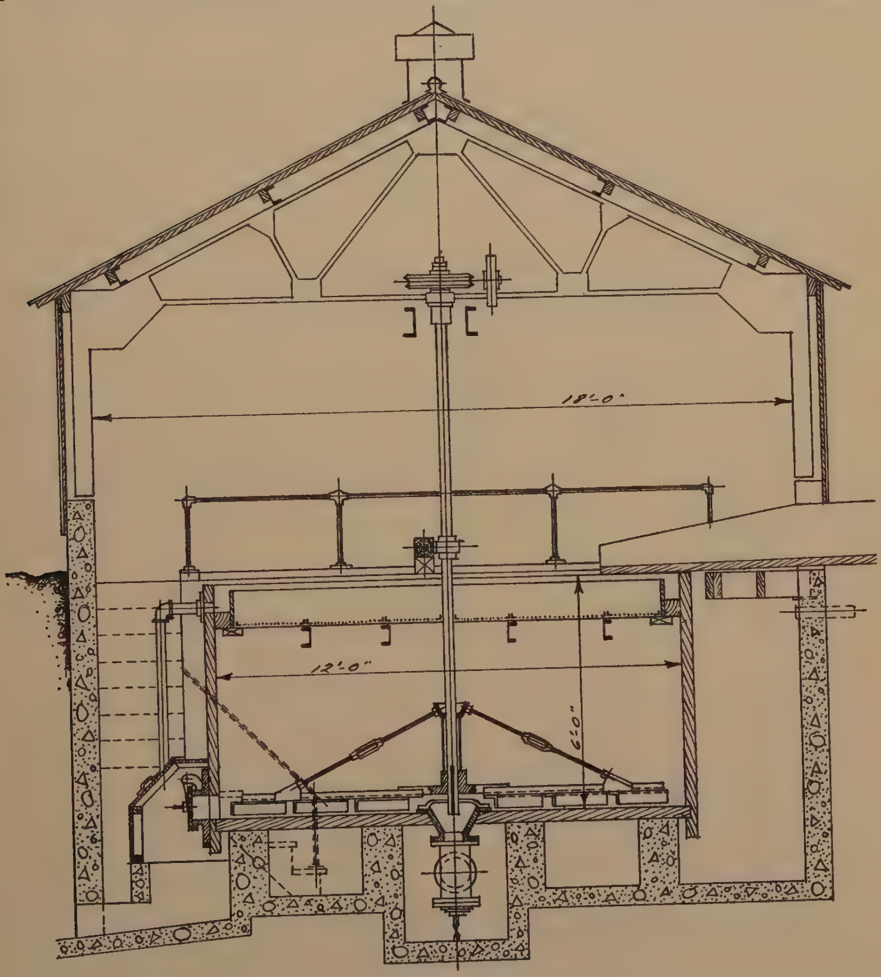


Fig. 1.—Sand house or conditioner.

Several years ago a few stopes in an isolated part of the mine were filled with sand tailings. Some years later this area was reopened for examination. The condition of the fill was so good that plans were made for experimental work with sand-filling of stopes in the main ore bodies.

Use of First Sand

Sand-filling of the Pierce ledge on the upper levels was begun in the summer of 1932. Settling of coarse waste fill had left open ground along the crest of the pitching anticline from the 200-ft. level to the 1200-ft. level. Three 4-in. diamond-drill holes were drilled from the surface into the top of Pierce stopes. Sand tailings were pumped from the cyanide sand plant to a portable dewatering cone discharging into one of these holes. This was successively moved to the other holes as each received all the sand it would take. This was completed in 1933. The amount of sand introduced was estimated at 114,000 tons. This experience with sand filling proved successful and economical and a number of things were learned about method and details of operation. Work was then begun to make possible sand-filling of stopes at deeper levels in the mine.

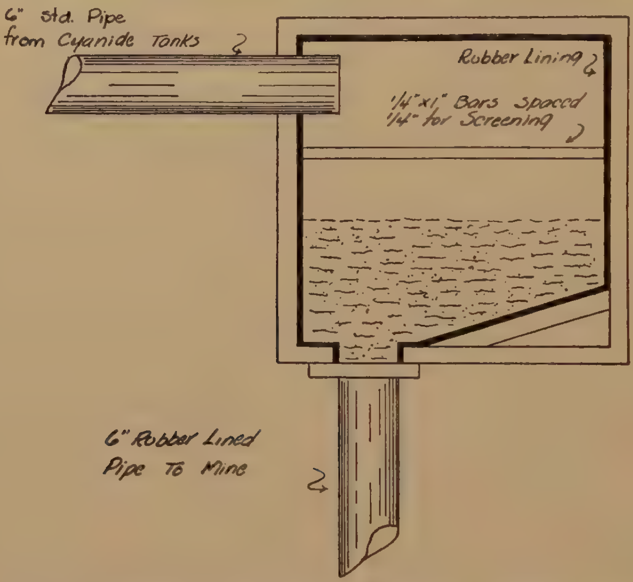


FIG. 2.—AIR-RELEASE BOX. SCALE: ⅜ INCH = APPROX. 1 FOOT.

For three years all of the sand available was poured into shrinkage stopes that had previously been filled with coarse fill. The sand filled the voids in the coarse fill, which increased the percentage of fill to at least 75 per cent. Three shrinkage stopes were filled with sand only, but a combination of coarse and sand fill is most desirable. The first sand-filling of pillar stopes was in January 1937. Now all stopes are so filled.

Each vat in the cyanide sand plant contains 700 tons of sand, which is discharged by sluicing. The mixture of sand and water goes to the sand house or conditioner (Fig. 1), where it is dewatered to the desired consistency for flow into the mine. Approximately 60 per cent solids has been found most satisfactory. At first, air locks in the mixture caused much hammering and some breakage in the sand line. This was overcome by breaking the line before it entered the mine. The mixture runs to an air-release box (Fig. 2) from the bottom of which the line passes into the mine.

In the early sand-filling operation, old compressed-air pipe was used to convey the sand from the cyanide plant to the stopes, and elsewhere. Some of this pipe has been in use in the mine for 35 years. Because of the highly abrasive nature of the sand, a great deal of trouble was caused by the sand cutting through the pipe in the shaft. When a leak occurred it meant that all work stopped in the shaft. After the leak was repaired, more time was lost in cleaning the sand out of the cages, particularly the safety dogs. The decision was reached to install 6-in. rubber-lined pipe for the main sand lines and use old 4-in. air pipe for the laterals.

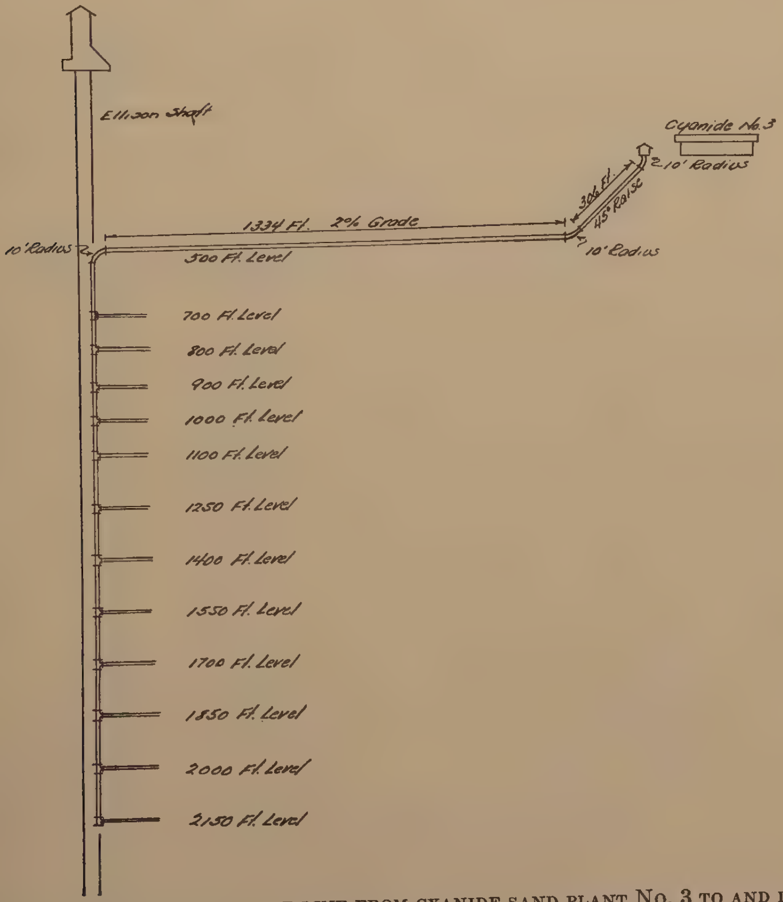


FIG. 3.—DIAGRAM OF MAIN PIPE LINE FROM CYANIDE SAND PLANT NO. 3 TO AND DOWN THE ELLISON SHAFT.

IMPROVED EQUIPMENT

Crosscuts and raises have been driven to permit gravity flow into the mine from each of the two cyanide sand plants. The installation of rubber-lined pipes for all mains was completed at the beginning of 1938. These mains are shown diagrammatically in Figs. 3 and 4. Throughout the shaft 6-in. rubber-lined pipe is used and a rubber-lined T is installed at each level. Fig. 4 shows a change in grade from 2 to 0.4 per cent, owing to the fact that the 0.4 per cent part of the pipe is laid in a regular haulage drift. The laterals leading off from the T's and across the levels are all 4-in. iron pipe lines, and their adaptations to the T's are accomplished by bell reducers. All deviations from a straight line are made by use of a long, sweep-type of elbow, in order to decrease resistance as much as possible and thus eliminate rapid wear on the interior of the pipe. Slide flange joints (Fig. 5) are used throughout the lined pipe. These flanges are drilled to accommodate 12 bolts, which draw the ends of the pipe together to make a tight compression joint. The manufacturers of the pipe have calculated that such a joint drawn up to make approximately a 25 per cent compression seal is adequate to withstand the maximum pressure in the line.

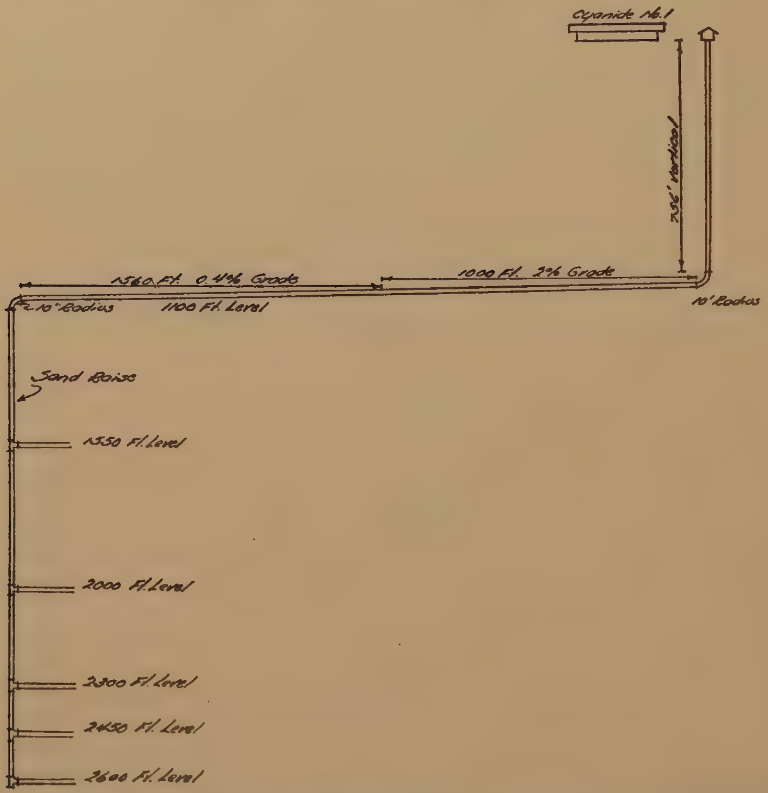


FIG. 4.—DIAGRAM OF MAIN PIPE LINE FROM CYANIDE SAND PLANT No. 1 TO 2600-FOOT LEVEL OF MINE.

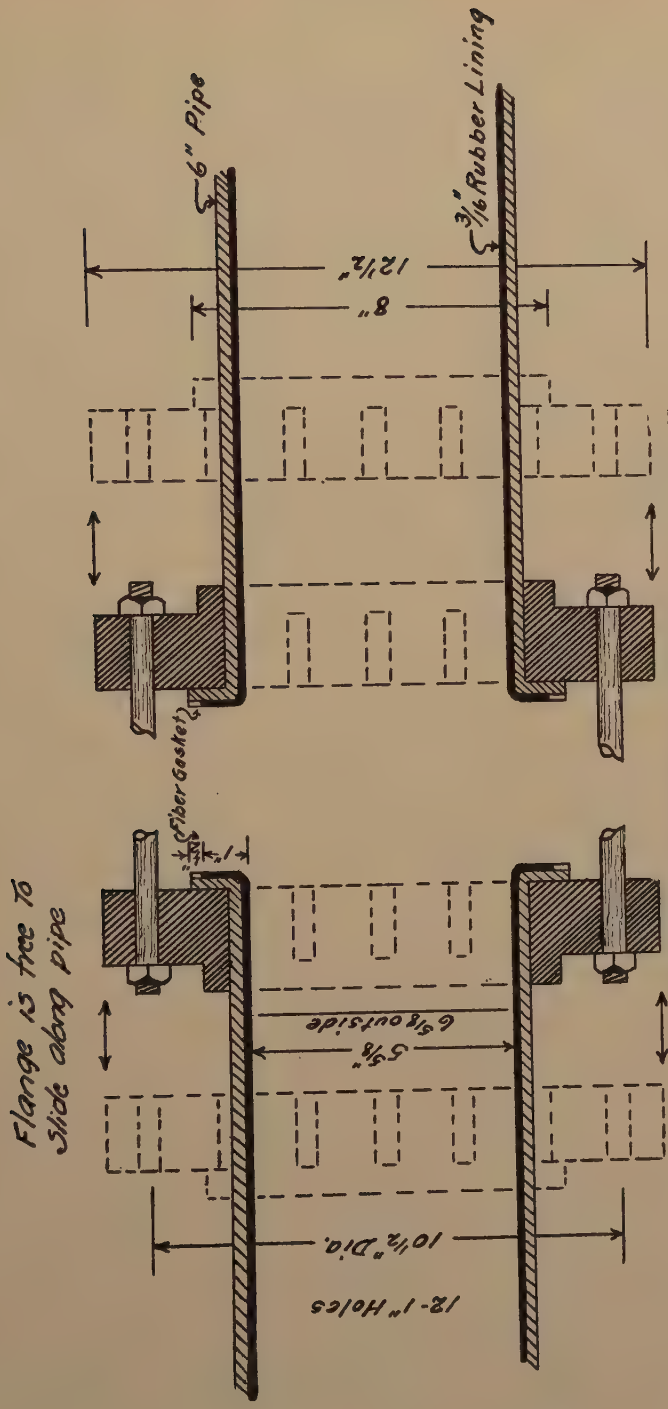


FIG. 5.—SECTION THROUGH RUBBER-LINED PIPE COUPLING.

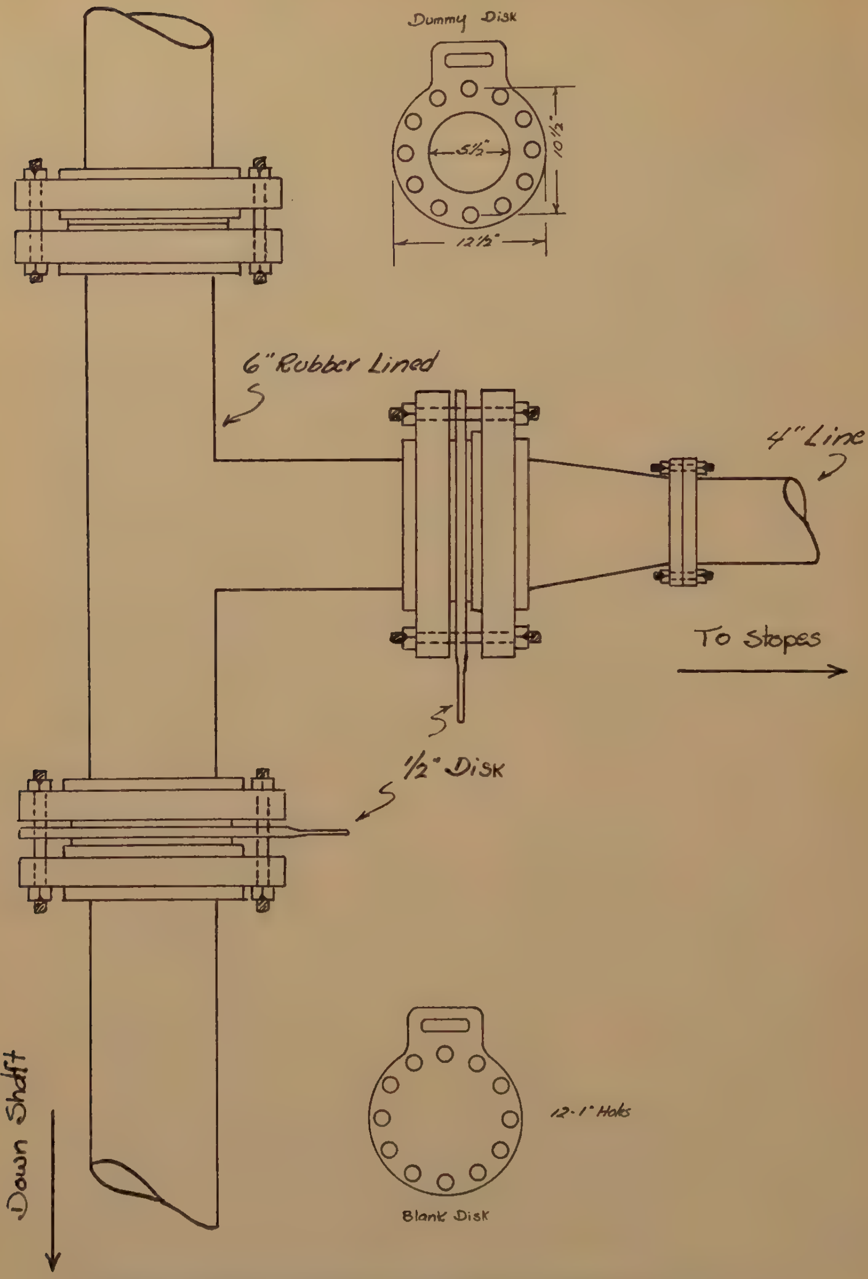


FIG. 6.—DIAGRAM SHOWING USE OF DISKS TO DIRECT FLOW OF SAND.

By use of blank, rubber-coated disks and dummy disks (Fig. 6) the mixture is directed as desired in its course down the shaft. As the pipe line is stationary in the shaft, the dummy must be used in connection with the blank disk. To turn the sand mixture off on to any lateral, the solid rubber-coated disk or blank is placed in the lower branch of the T, and the dummy disk, which is of the same thickness, having a 5½-in. opening in the center, is placed in the stem end of the T, where it is joined to the bell reducer. To by-pass a lateral, the respective disks are placed in the opposite locations. About 40 minutes is required to change a blank in the 6-in. lines and about 20 minutes in the 4-in. lines.

Probably the most excessive wear in the entire sand line is experienced at the T where the mixture is directed into the lateral. To overcome this difficulty, the rubber-lined T was made with long shanks so that when the blank rubber disk is placed in the lower shank a cushion of sand and water is formed above it on which a great deal of the abrasive action of the moving mixture is expended. Standard valves were originally used to direct the flow of the mixture, but the use of disks resulted in a substantial economy. Valves were costly and their life was short.

Mixture Used

The mixture used for filling averages about 60 per cent solids. It is controlled by a man stationed at the tank conditioner on the surface. Water flow into a tank is increased or diminished as the consistency of the mixture demands. The sand is graded through a 200-mesh screen with approximately 50 per cent of the solids having a size of minus 200 mesh. During the last 20 minutes of a run, the mixture is by-passed from the sand line to the creek, as during this period the mixture has a lower solid content than is desirable.

Three thousand tons of sand is available each 24 hr., of which 2100 tons can be utilized. Three underground crews are employed for sand-filling, each crew consisting of two men, who work a shift of 8 hr. under the direction of a day-shift foreman. Three runs of sand can be made daily, each run totaling about 700 tons of sand. Periodic tests are made on the surface and underground to determine the percentage of solids. If at any time it is desirable to stop the run, the underground man contacts the tank operator by telephone, and he in turn closes the tank-outlet valve.

Preparing Stopes

Shrinkage stopes (Fig. 7) at the Homestake vary in size. They are mined without the use of timber as support, and in some instances are carried up more than 150 ft. Before such a stope can be filled, all entrances must be walled off. During the earlier sand-fill work, cement walls were used entirely for these barriers, but experience has proved that a timber bulkhead properly built is more satisfactory, as it permits better drainage and is more flexible to ground movement if the latter should occur during the time of filling. Pitch timber, which is rejected for square-set work, is used for these bulkheads. To close up a stope, hitches are cut in the walls on each side of the drift or crosscut leading to the stope. The bulkhead (Fig. 8) is made in the shape of a broad arrowhead, the point directed toward the interior of the stope to be filled. Timbers, 12 by 12 in., are sawed to form an apex toward the stope and laid in a horizontal position. A heavy cement is mixed and forced in around the ends of these timbers, thus anchoring and sealing them into the wall hitches. This bulkhead is built up until there is a space of only 18 in. at the top. Using this opening as an entrance and exit, a man goes in and lines the inside of the partly formed barrier with burlap, leaving enough excess so that the lining can be pulled up over the last timber before it is fastened in place. After the horizontal timber is fastened in place, 12 by 12-in. timbers are placed in vertical positions conforming to the arrowhead shape of the horizontal timber. The spacing of timber throughout the entire bulkhead is approximately ¼ in., thus allowing ample drainage without loss of sand.

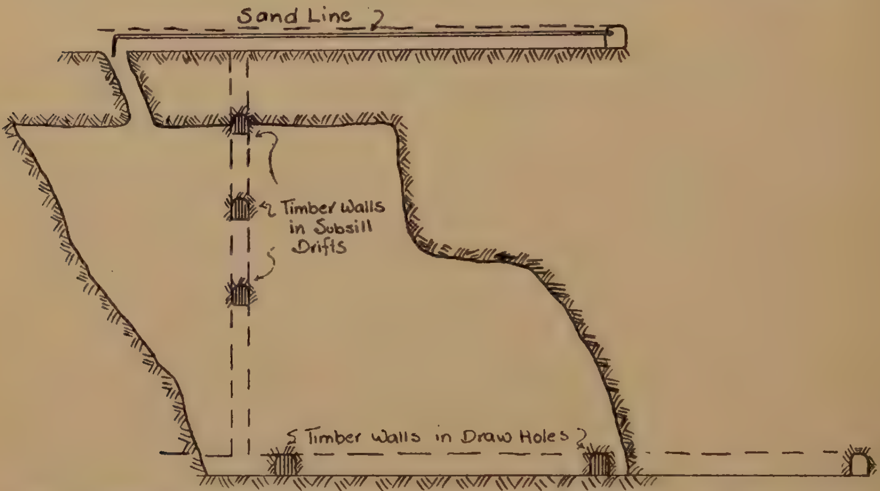
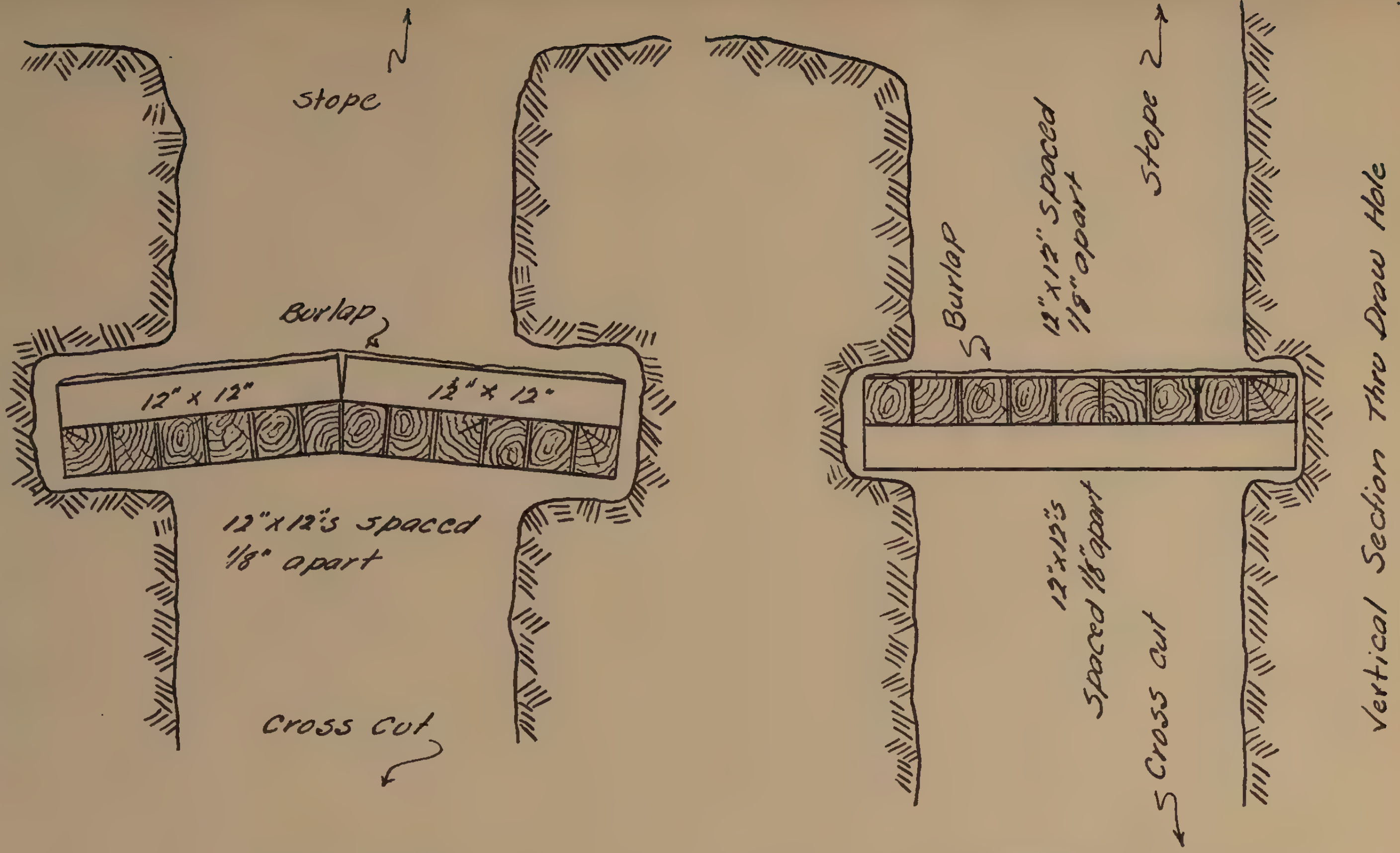


FIG. 7.—METHOD OF PREPARING SQUARE-SET STOPE FOR SAND FILL. SCALE ¾ INCH = 50 FEET.

When a span across a stope opening is unduly long, a concrete bulkhead is sometimes used. Such a wall is equipped with two or three valves spaced evenly from top to bottom.

The sand line is usually brought into the stope from an opening above. Thus, with the sand dropping down into the stope, a tight fill is secured to the very top of the stope. Force feeding from the bottom of the stope has been tried on several occasions. When such a method is used, it is



Plan of Draw Hole

Elevation

FIG. 8.—SECTION THROUGH SHRINKAGE STOPE SHOWING LOCATION OF WALLS OR DAMS PREPARATORY TO SAND FILLING.

necessary to have a shut-off valve inserted in the line. A large, strong core valve is utilized for this purpose and is placed as close to the end of the lines as possible. Immediately after a run is completed, this valve is closed, in order to prevent the fill already in the stope from backing out into the pipe line. It is not uncommon to have air-cushioning chambers formed in the "high spots" of the back of a stope when force feeding from below is used. This action prevents as tight a fill as would otherwise be possible if the sand were dropped in from an opening above the stope.

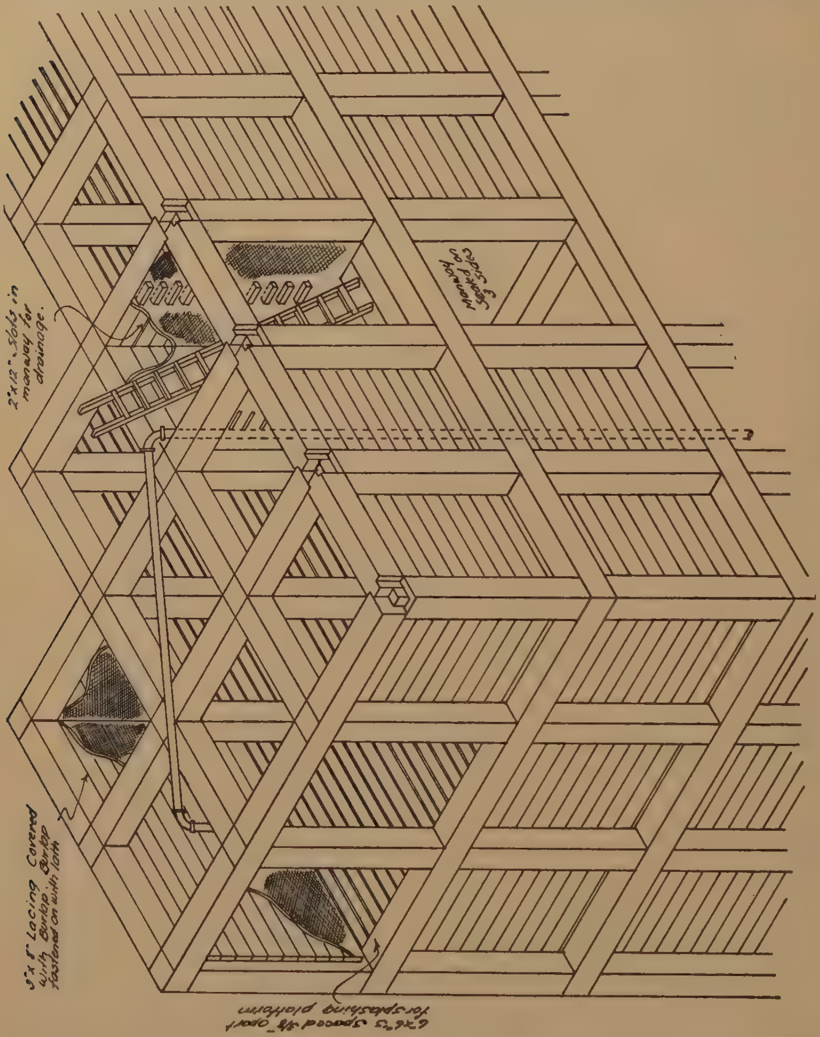

2"x12" Slots in manway for drainage.
Manway Lagged on 3 Sides
3"x8" Lacing Covered with Burlap. Burlap fastened on with lath
6"x6"s Spaced 4ft. apart for splashing platform

FIG. 9.—DETAILS OF WALLS OR DAMS. SCALE: ⅛ INCH = 1 FOOT.

The square-set stope method is used for the removal of all pillars of ore. The average timbered stope (Fig. 9) is seven sets long and three sets wide, each set being 6 by 6 by 8 ft. except the sill sets, for which 9-ft. posts are used. Access to and from the mine floors of such a stope is accomplished by means of a manway at the further end of the middle line of timber sets.

When one of these stopes is to be filled, it must be laced and sealed with burlap before it is ready for sand. One man is assigned to supervise this work preparatory to filling. The sides of the stope are laced with wood slabs 6 ft. long, 8 to 12 in. wide and 3 in. thick. The only opening left into the stope is the manway, which is laced off and made independent. This manway is used while mining the next stope. After lacing is finished, a 12-oz. burlap is used to line the interior of the laced sides. The purpose of the burlap is to give ample drainage for water and yet hold back the sand. An idea as to the efficiency of the burlap in accomplishing this can be gained from the fact that slab spacing of ¼ in. is permissible in lacing a stope. As the stope is laced and lined all floor laggings are removed.

The manway of the stope is used for the main drainage area (Fig. 9). As it is laced, each slab on the side facing the center of the stope has a 2 by 12-in. slot cut in it. Ample plugs covered with burlap are left to take care of these slots on each floor of the manway. The sand line is carried up the manway to the top floor and is arranged so that it will deliver the sand fill as far as possible from the manway, thus facilitating better drainage as the stope fills. An elbow is placed on the end of the line and turned in a downward position. A breaker box, for breaking and deflecting the concentrated stream of sand and water, is placed directly below the mouth of the elbow. No direct control valve is used in the sand line leading into the stope. Provisions are made for by-passing the mixture, in case of leakage in the stope, directly onto the sill, where barriers 3 ft. high are erected at the extremities of the stope. These barriers take care of the sand long enough to give the sand man time to have the run stopped.

The burlap used for lining is made in Czechoslovakia, and is purchased in bales of 15 bolts. Each bolt is 100 ft. long and 6 ft. wide. At present only the 12-oz. weight is used, but it is planned to experiment with a 10-oz. weight burlap and cotton cloth in the near future.

FILLING OPERATION

From the time a run is started until it is finished, a man is continually in the manway of the stope watching its progress. The water drains out through the slotted slabs previously mentioned. When the sand builds up and starts flowing out through a slot, it is plugged, and drainage is accomplished through the slotted slab directly above. The slabs used in the lacing are never over 12 in. wide; therefore during the process of filling there is never more than a foot of water in the stope. This process of plugging slots in turn continues until the stope is filled.

Three runs are usually required to fill a timbered stope. During the last run the section of the pipe line leading across to the breaker box is turned so that it will discharge its contents upward against the carpet, thus making a tight fill. The runs are usually made on successive days and each run is thoroughly set before the next run is emptied. A few hours after the stope has been filled it is thoroughly drained and set. Fifteen tons of sand is required to fill a sill-floor set; all other sets require 12 tons.

In filling a square-set stope that is inaccessible at the top, the sand line is brought in at the bottom of the stope and a riser put up to the top. Stopes have been successfully filled with as much as 1400 ft. of horizontal pipe on the level with a 75-ft. riser to the top of the stope.

More than two weeks is required to fill a stope with coarse waste, as compared to the three days required using sand fill.

Sand fill is available for all underground stopes. At the present time the sand line extends down to the 2150-ft. level and since a 100-ft. head is adequate for 300 ft. of lateral movement, the fill can be carried anywhere within the present stoping area.

Pipe Wear and Drainage

Two of the main problems encountered in sand filling are pipe wear and drainage. Satisfactory advancement toward the solution of both problems has been made. The rubber-lined pipe has thus far proved itself to be quite resistant to the abrasive wear of sand and it is believed that it will adequately serve for the conveyance of sand for a period of at least five years; 297,000 tons of sand passed through rubber-lined pipe into the mine during the last eleven months of the year 1938. Ordinary metal pipe lasted, in most cases, about three months; occasionally about six months. Burlap linings for stopes and bulkheads have already solved many drainage problems and the use of such linings has saved time and labor, owing to the elimination of the necessity for tight fits with all timber.

Trouble has been encountered in draining some sand-filled shrinkage stopes. A short time ago an experiment was tried in draining water out of a sand-filled shrinkage stope. The openings, with one exception, into the stope were bulkheaded as in the past. In the opening not bulkheaded a single square set was placed just inside the stope. This square set was lagged on three sides and covered with burlap as in the square-set stopes. As the sand level rose, additional sets were placed on top of the set in place. The sets were anchored to the solid rock as they were placed in position. This stope was filled and drained without any trouble and this method of draining water from shrinkage stopes will be used from now on.

When the time arrives to mine the pillar, this square-set passageway will be used as a fresh-air passage and manway.

Table 1 shows costs of sand-filling shrinkage and square-set stopes. These costs should be reduced in the future, because many errors made in sand-filling in the past will not be repeated.

TABLE 1.—*Cost of Sand-filling*[a]

Operation	Cost per Ton Filled
SHRINKAGE STOPES	
469,818 tons backfilled:	
Pipe lines:	
Supplies (pipe and pipe fittings)	$0.023
Labor (installation and maintenance)	0.019
Operating (running sand):	
Labor:	
Watching lines and draining stopes	0.030
Operating cones and sand pumps	0.011
Cleaning up sand and miscellaneous	0.007
Supplies	0.042
Development (500-ft. level, raise and drift)	0.013
Bulkheads:	
Labor	0.064
Supplies	0.030
Pumping sand water out of mine	0.037
Total cost	$0.276
SQUARE-SET STOPES FOR TWO MONTHS	
7,467 tons backfilled:	
Pipe lines	$0.042
Operating (running sand)	0.223
Development (raise and drift)	0.013
Lacing stopes:	
Labor	0.291
Supplies	0.287
Pumping sand water out of mine	0.037
Total cost	$0.893

[a] Pipe supplies and development costs have been deferred over their useful time. The above costs were obtained previous to the present rubber-lined pipe installation.

Splash boards are used because sand being deposited in the timbered stopes under high pressure had a tendency to cut the supporting timber in two, thereby causing the stope to cave before filling was completed. Since the use of splash boards was begun no cutting of timber has taken place.

ADVANTAGES AND DISADVANTAGES

The benefits of sand filling are of much greater importance than the disadvantages. These benefits are as follows:

1. Surface subsidence has been very greatly retarded.
2. Openings into the top of pillars now being mined are easily maintained at greatly reduced costs.
3. Ventilation is greatly improved both because of the maintenance of openings and the fact that air is not dissipated through the fill, as with coarse fill.
4. Working conditions are safe where all stopes in a pillar are closely filled.
5. Increased size of stoping sections is possible and the output per man thereby increased.
6. Economy in timber because it is not necessary to use relief posts, cribs, diagonals, etc.
7. Fire hazard is reduced because the timber does not dry out as rapidly.

Sand filling has the following disadvantages:

1. Mine drainage and pumping costs are higher. All drainage ditches must have adequate sand traps to prevent the highly abrasive sand from reaching the pumps. The quantity of water to be pumped is increased.
2. Openings that are made into sand-fill areas sometimes encounter air with a decided deficiency of oxygen. This is easily overcome by the use of fan and ventpipe.
3. Sand spills are almost unavoidable and the task of keeping the mine clean is increased.

ACKNOWLEDGMENTS

Before sand filling started in the Homestake mine, a great deal of helpful information on sand filling was obtained from the staff of the Anaconda Copper Co., Butte, Mont., and the staff of the Matahambre mine, Cuba.*

DISCUSSION

(Benjamin F. Tillson presiding)

C. M. HAIGHT,† Franklin, N. J.—The pneumatic method was used at Franklin to place sand fill where haulage could not be used. Mill tailings were blown from a tank like those in use at the Champion copper mine in Michigan, through steel pipes 3 in. in diameter.

---

* The development at the Matahambre mine has been described by G. I. Richert in *Information Circular* 6145 of the U. S. Bureau of Mines, and in articles in *Engineering and Mining Journal* (March 1929) and *Engineering and Mining World* (December 1931).—Ed.

† Mine Superintendent, New Jersey Zinc Co.

The sand was damp from its origin and also from its contact with the mine air and with water; it was soon found that the addition of water to the sand as it was placed in the tank resulted in less plugging of the pipes. Wear was greatest in the pipes at bends, so hose, 3½-in. diameter, was tried at these places, and lasted enough longer than the pipe to justify the increased first cost.

The present placement of sand is around timber or steel sets (caps 5 ft. inside, legs 7 to 9 ft. long) in large drifts, some 15 by 15 ft. or larger, so as to completely fill the drifts. Plenty of water is used, to facilitate transportation through the pipes and cause a compact fill. Pipe 6 in. in diameter is used with a 25-ft. length of 6-in. hose at the discharge end to help the distribution of the sand. The larger pipe gives better results than the 3-in. pipe.

Where the headroom was too low for the use of the tank, a pneumatic concrete placer was used to blow the sand. The longest distance at which fill has been placed was 500 ft. from the tank.

G. S. Rice,* Alexandria, Va.—Was the sand filling done primarily to permit better extraction of ore or was it done to prevent or lessen overlying ground movement?

C. F. Jackson,† Washington, D. C.—What is the screen size of the sand used for filling at Homestake?

G. N. Bjorge,‡ Lead, S. D.—The sand contains about 45 per cent of minus 200-mesh material (see Table 2).

C. F. Jackson.—It is surprising that sand of this degree of fineness will drain so rapidly, and the reason undoubtedly is that it is a classified material. The question of drainage is important, because if the fill will not drain, water is apt to accumulate in such large quantities that if suddenly released by ground movement or other cause disaster may result. At the Hollinger mine, where dry sand and gravel fill is procured from an open pit in glacial material, laboratory percolation tests are made on the sands once a week. If water will not percolate through the sand in a vertical 8-in. pipe, 10 ft. high, at the rate of 6 in. per hour or faster, it is considered unsuitable for backfill, since it may cause an accumulation of drilling water.

Although the amount of cyanide retained in leached sands is much less than popularly believed, and is probably negligible, I should like to ask whether any treatment other than the regular water wash has been necessary at Homestake to ensure against cyanide poisoning in the mine. I have been told that in South Africa cyanide tailings have been treated with copper sulphate to kill the cyanide before they are used for backfill.

G. N. Bjorge.—No special treatment is required at Homestake.

G. Sherman,* New York, N. Y.—What was the capacity of a 6-in. pipe in tons of tailings delivered?

G. N. Bjorge.—About 700 tons have been put into a stope in 4 hours.

G. Sherman.—Is the wear on the rubber lining in a horizontal section of pipe greater on the bottom third of its circumference?

G. N. Bjorge.—No such wear has been observed to date.

---

* Consulting Engineer.

† Chief Engineer, Mining Division, U. S. Bureau of Mines.

‡ General Manager, Homestake Mining Co.

G. Sherman.—Do the costs given in the paper apply to the tons of tailings delivered into the stopes or to the tons of ore replaced?

G. N. Bjorge.—The costs were based on the tons of tailings and would be less per ton of ore mined.

L. Eaton,* Milton, Mass.—Some years ago I made a careful study of sand filling through pipes. I think Mr. Cromwell will corroborate me in the statement that there is a definite relationship between the size of the pipe and the amount of sand to be transported. The pipe must not be too large—too large a pipe clogs easily, and too small a pipe has excessive wear.

One of the best applications of sand filling that I have seen was at the Hodbarrow mine, at Millom, on the west coast of England. The ore extended under the Irish Sea, and it was necessary to prevent subsidence from going beyond the sea wall. The ore was a soft hematite, which was mined by bottom slicing with sand fill. The fill was carried in iron pipes, into which small jets of compressed air were admitted at some of the turns, and care was necessary to prevent the jet from impinging on the wall of the pipe. There was no difficulty in elevating the mixture of sand and water to the stopes or in making complete fills. The lagging was lined with burlap before the sand was placed, and there was little trouble with drainage or contamination of the ore.

R. H. Cromwell,* East Orange, N. J.—A discussion of the paper by Mr. Ross would not be complete without mention of the pioneering work done in sand filling at Matahambre, Cuba. Of course, this method of filling stopes has been used for a long time in South Africa and on the Continent, but I believe that Matahambre was the first to work out the details and use this method in a large-tonnage mine operation in this part of the world. The Matahambre practice was written up in the *Engineering and Mining Journal* in 1929, and was followed by an article in the same journal in 1931, by D. D. Homer, on the use of a particular rubber lining for the sand-fill pipe lines. This contributed greatly to success in delivering the sand fill continuously where it belonged in the stopes, and not in the mine drifts and raises as often happened in the early days of sand fill at Matahambre. The rubber lining of this 3-in. pipe was ¼ in. thick, rigidly vulcanized, so that it adhered to the inside of the iron pipe. The rubber-lined pipe could then be bent cold without materially disturbing the lining. It was found by experiment that the thickness of the rubber lining could not be reduced to ⅛ inch.

At Homestake, rubber-lined disks and T-connections are employed to distribute the sand from the main vertical line to the different main levels; while at Matahambre, to cut down the wear at these points, sweeping bends of as large radius as the drifts would permit were used in the pipe lines. The flow of sand-fill pulp was controlled at the surface when it was necessary to switch the sand fill from one level to another.

The main point about filling with mill tailings is to take out the slimes and colloidal material so that the sands will drain in the stopes, and to closely control the amount of solids in the pulp going underground. Woe betide anyone who attempts to fill stopes with mill tailings without complete removal of the slimes and colloidal material!

Most mine operators are prejudiced against "sand fill" because water must be returned to the mine, thereby increasing the pumping expense. The delivery of 2100 tons of fill per day to the stopes at Homestake in a pulp containing 60 per cent solids means an increase of approximately 90 gal. of water per minute to be pumped. The use of sand filling at Homestake will probably lead to the introduction of the method to other large mines where conditions are favorable.

---

* Consulting Engineer.

A. J. M. Ross (author's reply).—Specially built 4-in. (inside diameter) rubber hose is being used successfully at Homestake for bends on the laterals. A flanged end connects easily into the 4-in. pipe lines. Time, labor and headroom are saved by using the rubber hose instead of short nipples, special bends and fittings.

Sand filling serves the dual purpose of both speeding up extraction of ore and lessening subsidence.

Screen analyses are given in Table 2 for cyanide sand plants Nos. 1 and 3. The product from the latter plant works better and is preferred. The difference lies primarily in the percentage of solids—the latter being higher (62 per cent). The average percentage of solids of the cyanide No. 1 fraction of sand pulp discharged to the mine is 53; from No. 3 plant is 62. (These sizings are averages from daily samples during 1937, 1938 and 1939.)

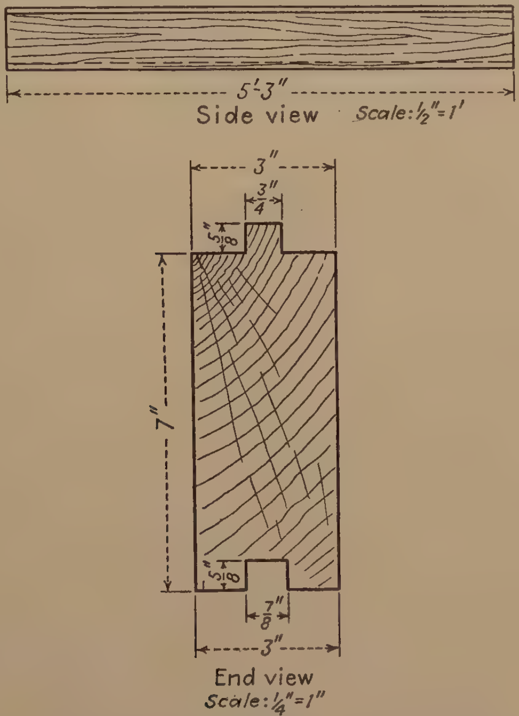


FIG. 10.—WOOD SLABS FOR SAND FILLING.

Early in 1939, owing to the effect of the European situation on the jute market, and in turn on burlap, experiments were undertaken in an effort to find a satisfactory substitute. The cotton manufacturers produced one but at a price about 25 per cent higher than the governing price for burlap.

Experiments were then made with new types of stope lacing, both shiplap and tongue and groove plank being successfully employed. The latter proved easier to handle and made a better joint. Three-inch plank with a 7-in. face is grooved to a width of ⅞ in. and a depth of ⅝ in. and tongued on the opposite edge to a width of ¾ in. and a depth of ⅝ in (Fig. 10). By carefully fitting this lacing, about 75 per cent of the burlap was eliminated, as it was no longer necessary to face the lacing with it.

TABLE 2.—*Screen Analyses of Sand*

Mesh	Size, Per Cent	
	No. 1 Plant	No. 3 Plant
+50	1	0
+80	8	3
+100	9	6
+150	21	19
+200	25	28
−200	36	44

Subsequently, by the use of a gunite machine, the corners and edges of the lacing, where they contacted the stope wall, were sealed in with cement and thus the two new steps in combination almost entirely eliminated the use of burlap and at the same time sped up the work of preparing a stope for filling. A stope can now be laced and filled in about one-half the time formerly required. Obviously the tongue and groove lacing costs more than ordinary slabs and there is the additional expense of guniting, but on the other hand burlap is practically eliminated and the speed of the filling cycle is doubled, which in itself is highly important.

Progress of "Big Blasting" at Climax

By F. S. McNicholas,* Member A.I.M.E.

(New York Meeting, February, 1938)

In the first big blast at Climax, a "loop back" (three-wire system) was used (Fig. 1), with the idea of securing a wiring system that would give to all series the same amount of current. Single-phase, 60-cycle, 13,800-volt power was furnished to the blasting transformers, each of which has a rated capacity of 600 amp. at 480 volts. Two blasting leads were taken from these transformers, giving a blasting current of single phase, 60 cycle, 480 volts, with an instantaneous capacity of more than 600 amp. The blasting switch was on the secondary leads from the transformers, to avoid lag in energizing them.

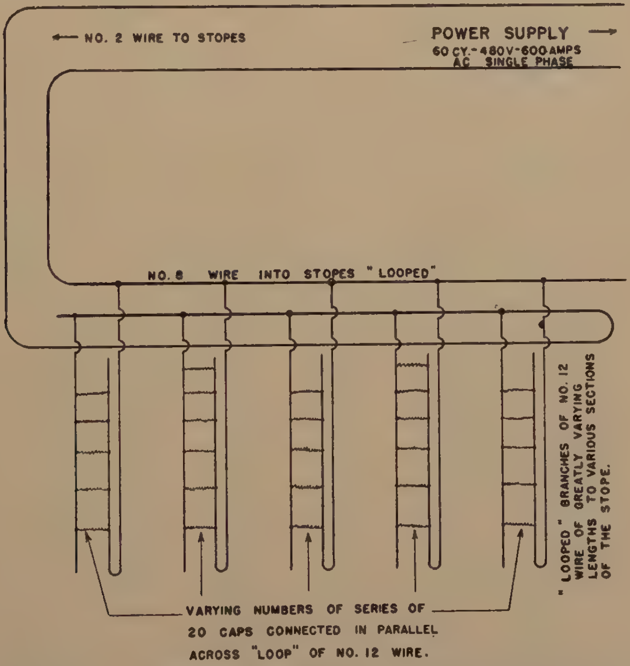

← NO. 2 WIRE TO STOPES
POWER SUPPLY →
60 CY.-480V-600 AMPS
AC SINGLE PHASE
NO. 8 WIRE INTO STOPES "LOOPED"
"LOOPED" BRANCHES OF NO. 12 WIRE OF GREATLY VARYING LENGTHS TO VARIOUS SECTIONS OF THE STOPE.
VARYING NUMBERS OF SERIES OF 20 CAPS CONNECTED IN PARALLEL ACROSS "LOOP" OF NO. 12 WIRE.

Fig. 1.—Loop back. A three-wire system with branches varying in length.

Manuscript received at the office of the Institute Jan. 10; revised June 28, 1938
Issued as T.P. 936 in Mining Technology, September 1938.

* Assistant General Superintendent, Climax Molybdenum Co., Climax, Colorado.

Series of 20 instantaneous and delay electric blasting caps were connected in parallel to the 12-gauge wire branch "loop" in the stopes.

Any series less than 20 caps was completed to 20 by insertion of the required number of regular delay caps (called dummies).

In several blasts some circuits, or portion of circuits, were found unexploded and a study begun to determine the reason. In No. 18 shot, where 11,316 holes and 24 powder tees were blasted, an accurate record of wiring was kept and an attempt was made to measure current and power supplied for the shot. The graphic wattmeter installed was too sluggish to give a reading of power. The graphic ammeter installed gave a reading that, after applying the usual correction, appeared to be approximately correct.

The ohmic resistance of feeder, branch lines, and caps was calculated and the theoretical amount of current required, based upon the ohmic resistance only, was calculated. This proved to be 59 per cent higher than the current furnished, showing that the impedance of the circuit was greater than the calculated resistance. An investigation of "inductive reactance" proved that in the main leads (consisting of two No. 2 gauge wires, 2050 ft. long, spaced on about 18-in. centers) this is a matter of importance and increased the impedance from the ohmic resistance of 0.6412 ohms to 0.806 ohms. "Inductive reactance" in the stope lines was negligible.

The Du Pont Powder Co. was asked to determine the minimum current required to perfectly detonate series of 10 and 30 caps of D-50 delays and No. 6 instantaneous blasting caps, all having 18-ft. lead wires. The results are shown in Table 1, indicating that the minimum current required is 0.7 amp. per series of 20 delay caps with 18-ft. leads. Since 602 series were blasted, the current required was 602 × 0.7 = 421.4 amp. From the graphic ammeter we secured a reading of 277 amp. furnished, which would seem to indicate that insufficient power was furnished or a "graded series" effect was secured.

TABLE 1.—*Current Required to Detonate Caps*

Supply Voltage	Amperes Required		
	D-5 Delays		Instantaneous
	10 Caps in Series	30 Caps in Series	No. 6 E. B. C., 30 in Series
120		0.75	
450	0.65	0.75	0.60

From examination of resistances of various branches, we find a variation of from 1.328 to 18.22 ohms, which is sufficient to give a "graded series" effect. A "graded series" effect is secured when the resistances of various branches vary enough to cause sufficient current to flow through the branch of least resistance to detonate it ahead of those of greater resistance.

Table 1 shows that the current required to explode instantaneous caps in series of 30 is 0.6 amp., and of delays is 0.75 amp. It is evident that instantaneous caps should not be used with delays, as they may detonate before the delay bridges are fused and break the circuit. This effect would be accentuated where large and small branches are used. The instantaneous caps in small branches will explode first and perhaps break the circuit of the larger branches. This is particularly true when available current approaches the minimum needed for explosion.

The following conclusions were drawn from this study of No. 18 shot:

1. Instantaneous caps should not be used with delay caps in future blasts.
2. An exact measurement of current supplied would be secured in a subsequent blast by the use of an oscillograph.
3. An exact measurement of resistances would be secured at a subsequent blast to check against the calculated resistance.
4. A study of "graded series" effect by the use of the oscillograph would be made.
5. "Inductive reactance" of main feeder lines would be considered, as well as ohmic resistance.
6. The "loop-back" system with branches of *varying length* and *varying number of series in a branch* is *not totally balanced.* It is balanced to the extent that all branch lines have equal potential across them, but this need not be a desirable feature, since with varying lengths of branch lines and varying numbers of series of caps in each branch the potential across the branch lines must vary if equal current is to be supplied to each series of caps.

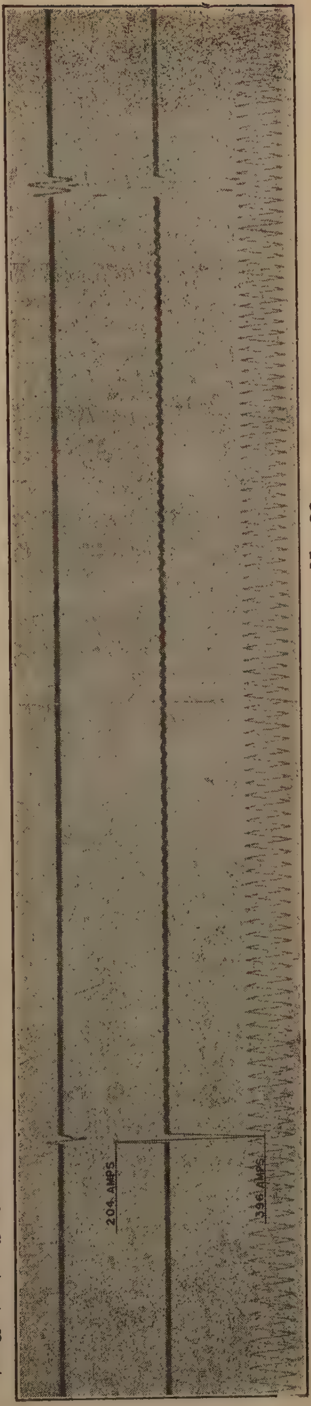


Fig. 2.—Oscillogram of Blast No. 22.

In blast No. 22, conclusions 2 and 3 were studied. The Public Service Company of Colorado cooperated by supplying an oscillograph and operators. The Hercules Powder Co. took various physical measurements of resistances. A record of wiring was kept.

In this blast, no instantaneous caps were used. In all other respects it was the same as blast No. 18.

Fig. 2, an oscillogram (all graphs are in duplicate as the pictures were taken on the two elements of the oscillograph), shows on the left side, middle line, the maximum current supplied of 396 amp., the duration of current of about 0.01 sec., thence reading the graph to the right, a slight leakage of current for 80 cycles, or about 1⅓ sec., then a disruption of the circuit by the detonation of the first delays. The lower graph is that of the 60-cycle voltage used as a timing wave.

Fig. 3 shows the various branch lines of No. 12 wire "loop" through the stopes to be blasted.

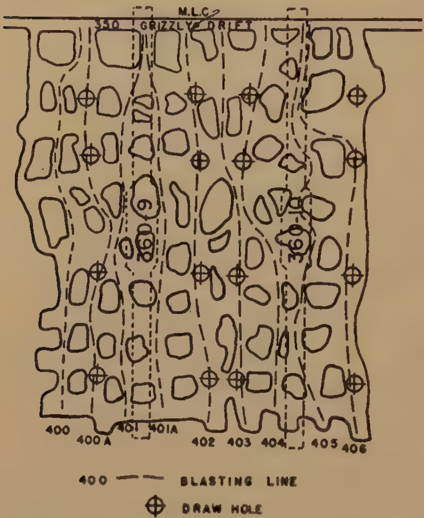


Fig. 3.—Branch lines of No. 12 wire loop through stopes.

FORMULAS

The blasting current is single-phase, 60-cycle alternating current, and the load is entirely a resistance load, therefore Ohm's law, $E = IR$, where E is the voltage, I the current in amperes and R the resistance in ohms, may be used for calculating all requirements, except feeder lines, which have considerable inductive reactance. Inductive reactance in stope and branch lines is so small as to be negligible.

The following formulas are approximate, as inductive reactance has been neglected in all circuits except the main feed line; also, other approximations are used, but these should be sufficiently accurate for ordinary calculations.

R_{cs} indicates resistance of a single "series" of caps; R_{tc} resistance of the total number of "series" connected to one branch line:

$$R_{tc} = \frac{1}{\dfrac{1}{R_{cs1}} + \dfrac{1}{R_{cs2}} + \dfrac{1}{R_{cs3}} \text{------}}$$

R_{blw} indicates wire resistance of the branch line. In a three-wire system, this is the wire resistance of twice the length of the branch line.

For a two-wire system, this is the wire resistance of twice the distance from the stope leads to the load center (*LC* on Fig. 7).

R_{bl} indicates resistance of one complete branch line, including "series" resistance (R_{tc}).

$$R_{bl} = R_{tc} + R_{blw}$$

R_{tbl} indicates total resistance of all the branch lines in one stope.

$$R_{tbl} = \frac{1}{\frac{1}{R_{bl1}} + \frac{1}{R_{bl2}} + \frac{1}{R_{bl3}} \text{------}}$$

R_{slw} indicates wire resistance of the stope leads. In a three-wire system, this is the wire resistance of twice the length of the stope line. For a two-wire system, it is the wire resistance of twice the distance from the main power leads to the load center of the stope.

R_{sl} indicates total stope resistance of one stope.

$$R_{sl} = R_{slw} + R_{tbl}$$

R_{tsl} indicates total resistance of all stopes.

$$R_{tsl} = \frac{1}{\frac{1}{R_{sl1}} + \frac{1}{R_{sl2}} + \frac{1}{R_{sl3}} \text{------}}$$

R_{ml} indicates wire resistance of main leads. In a three-wire system, this is the wire resistance of twice the length of the main lead. For a two-wire system, it is the wire resistance from the power supply point to the main load center (*MLC* on Fig. 3).

R_t indicates total resistance of entire system.

$$R_t = R_{ml} + R_{tsl}$$

X_{ml} indicates inductive reactance of main leads. For a three-wire system assume total reactance between two wires for total length of main lead; for a two-wire system use reactance for main leads for the distance from the power supply to the main load center (*MLC*).

Z_t indicates total impedance of the system.

$$Z_t = \sqrt{(R_t)^2 + (X_{ml})^2}$$

CALCULATION OF REQUIREMENTS, BLAST NO. 22

Resistance of one cap = 1.803 ohms. $R_{cs} = 1.803 \times 20 = 36.06$. Since all "series" were identical, Table 2 may be used to determine R_{tbl} and show the comparison of calculated values with measured value.

Table 2.—*Calculation of Requirements, Blast No. 22*

Branch Line No.	R_{blw}	No. of Series	R_{tc}	R_{bl}		$\frac{1}{R_{bl}}$	
				Calculated	Measured	Calculated	Measured
400	0.715	36	1.0	1.715	1.34	0.584	0.746
400 A	0.715	25	1.44	2.155	2.18	0.465	0.458
401	0.715	23	1.56	2.275	2.35	0.440	0.425
401 A	0.715	29	1.24	1.955	1.49	0.512	0.670
402	0.715	30	1.20	1.915	1.66	0.522	0.602
403	0.715	31	1.16	1.875	1.78	0.533	0.562
404	0.715	14	2.57	3.285	2.65	0.302	0.377
405	0.715	24	1.50	2.215	2.10	0.452	0.476
406	0.715	11	3.27	3.985	2.90	0.251	0.345
						$\frac{1}{R_{tbl}} = 4.061$	4.661
						$R_{tbl} = 0.246$	0.214

The branch lines were 225 ft. long made up of No. 12 copper wire "looped." There were nine branch lines. Resistance of No. 12 wire at mine temperature is 0.001588 ohm, hence:

$$R_{blw} = 2 \times 225 \times 0.001588 = 0.715$$

Stope loop was 150 ft. long of No. 2 copper wire with a resistance of 0.000156.

$R_{slw} = 2 \times 150 \times 0.000156 = 0.047$ ohm.

$R_{sl} = 0.047 + 0.246 = 0.293$ ohm.

Since there was only one stope lead, $R_{sl} = R_{tsl} = 0.293$ ohm.

The main leads were also No. 2 copper 930 ft. long.

$R_{ml} = 2 \times 930 \times 0.000156 = 0.29$ ohm.

$R_t = 0.293 + 0.29 = 0.583$ ohm.

Reactance of one conductor of single circuit of No. 2 wire with 18-in. spacing was 0.000119 ohm, or 0.000238 ohm for both wires of single-phase circuit.

$X_t = 0.000238 \times 930 = 0.221$ ohm.

$Z_t = \sqrt{(0.583)^2 + (0.221)^2} = 0.622$ ohm.

I = *current required* = 223 series of 20 caps × 0.7 amp. × factor of safety of 1.5 = 234 amp.

E = *voltage required* = 234 × 0.622 × factor of safety of 2.0 = 291 volts.

Actually Delivered:

I = 396 amp. crest value, or 396 × 0.707 = 280 amp. effective.

E = 480 volts.

$$E = IZ \qquad Z = \frac{E}{I} = \frac{480}{280} = 1.71 \text{ ohms.}$$

Comparison

	Actual	Calculated	Measured	Ratio Actual to Calculated
Total impedance, ohms	1.71	0.622	0.97	2.75
Crest current, amp	396			
Effective current, amp	280			
Effective voltage, volts	480	304		

The large difference between the calculated impedance and the actual is probably accounted for by poor wire connections. In Table 2

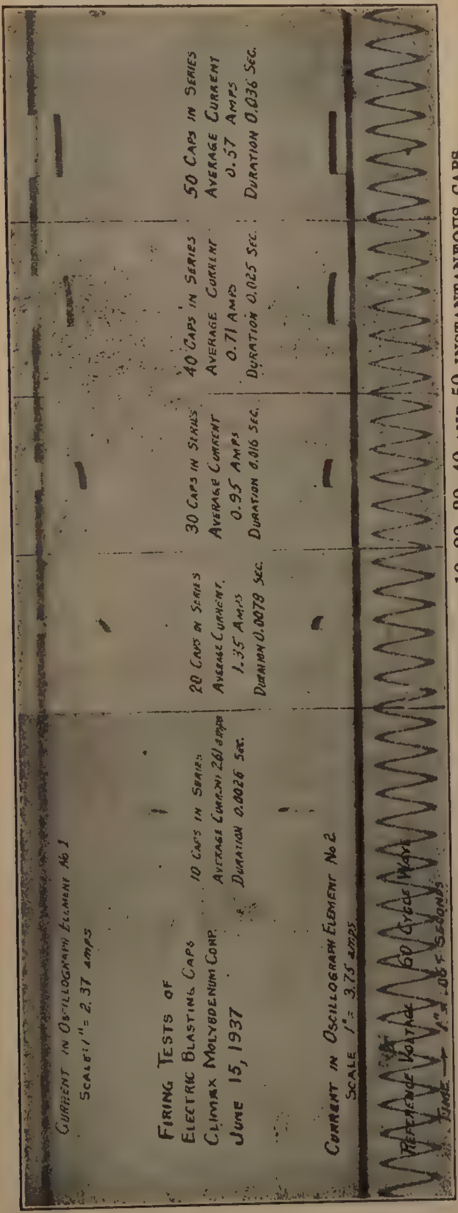


FIG. 4.—RECORD OF EXPLOSION OF SERIES OF 10, 20, 30, 40 AND 50 INSTANTANEOUS CAPS.

there is a very close check, which would indicate that the great difference in impedance is due to *poor connections outside* of the stopes.

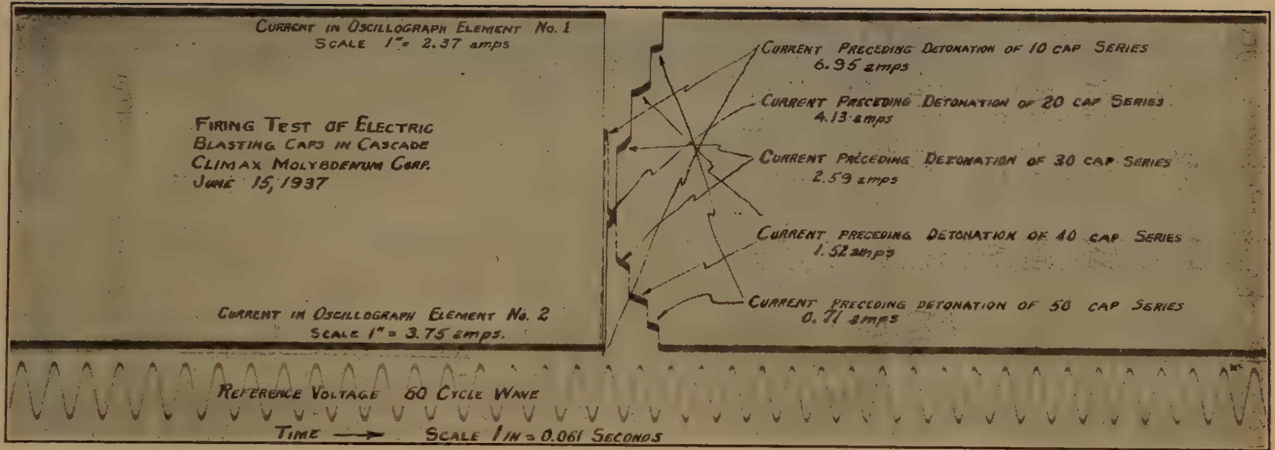


FIG. 5.—RECORD OF EXPLOSION OF SERIES OF 10, 20, 30, 40 AND 50 INSTANTANEOUS CAPS CONNECTED IN PARALLEL.

To Determine "Graded Series" Effect

To endeavor to determine "graded series" effect, direct current was used at a voltage of 63, which was the estimated voltage required to furnish the minimum firing current of 0.7 amp. to a series of 50 instantaneous electric blasting caps with 18-ft. lead wires connected in series. Individual series of 10, 20, 30, 40 and 50 instantaneous caps were exploded, giving the graph of Fig. 4, which is self-explanatory. Series of 10, 20, 30, 40, and 50 instantaneous caps were then connected in parallel in a blasting line and exploded, giving the graph of Fig. 5. Fig. 6 shows the data secured from Fig. 5, plotted to scale.

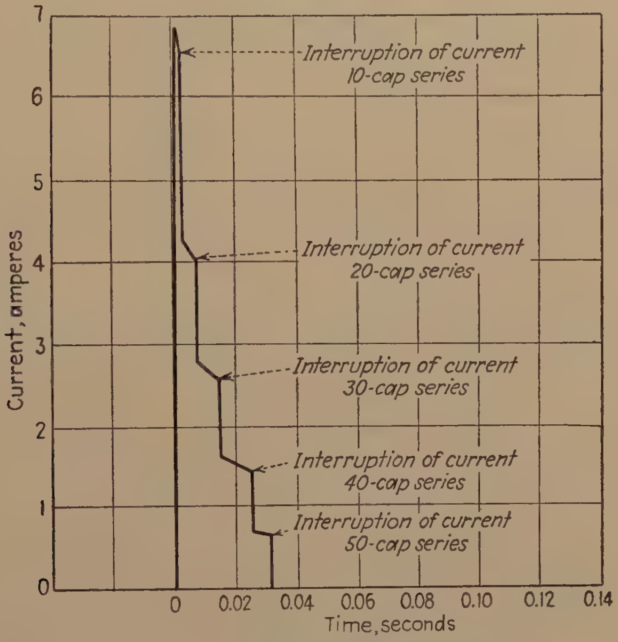


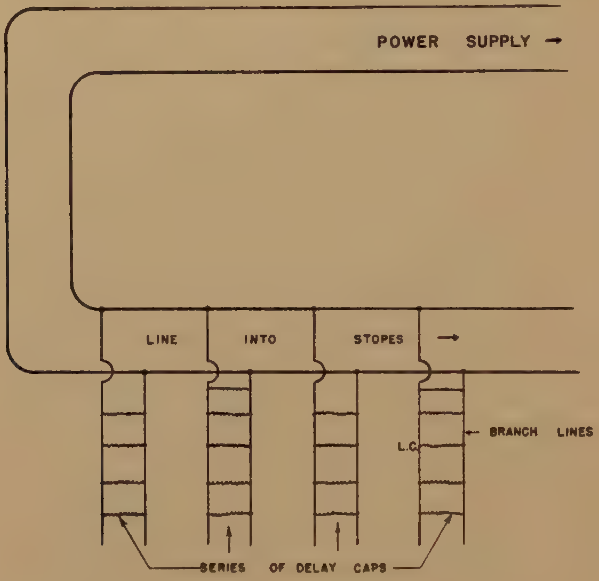
Fig. 6.—Data of Fig. 5 plotted to scale.

It is evident from Fig. 5 that a "graded series" effect has been secured by the variable resistance in each cap series. The duration of current for the various series varied from 0.003 sec. for the 10-cap series to 0.031 sec. for the 50-cap series. Since one cap in the 50-cap series failed to explode, it is indicated that the current of 0.57 amp. is slightly low to properly explode 50 caps.

Where alternating current is used, a greater current must be supplied to allow for the variable point on the cycle wave and inductive reactance.

Final conclusions drawn from study of No. 18 shot, No. 22 shot, and "graded series" effect are:

1. Do not use instantaneous electric blasting caps with delays.
2. Study of the "loop back" (three-wire system), shows that where branch lines vary in length and the number of series connected in these branches vary greatly, the effect of the loop back to secure equal current to each series is nullified.
3. From Fig. 6, it is seen that the duration of current for firing 50 caps in series is 0.031 sec. Fig. 2 shows that the interval between ignition of delay caps and detonation of first delays is 1.33 sec. It is evident, if instantaneous caps or zero delays are not used, that any arrangement of delay caps (up to 50 in series) will be ignited before detonation can take place to disrupt circuits.



ANY NUMBER OF SERIES IN BRANCH
ANY NUMBER OF CAPS IN SERIES UP TO 50
ANY COMBINATION OF CAPS IN SERIES
(WITHIN THE LIMITS OF AVAILABLE POWER)

CAP LEAD WIRES MAY BE CUT TO MAKE SNUG CONNECTIONS.

FIG. 7.—TWO-WIRE SYSTEM WITH BRANCHES AND LADDER HOOKUP.

4. Inductive reactance in main lines should be considered.
5. A factor of safety of not less than 2 should be used in calculating resistance.
6. Allow a factor of safety of 50 per cent in estimating current requirements.

RECOMMENDATIONS

From these conclusions, the following recommendations for blasting are made:

Do not use instantaneous caps with delay caps. Use no zero delays.

If this rule is followed, there are two classes of blasting big rounds: (1) blasting with delay caps (excepting zero delays); (2) blasting with instantaneous caps.

Case 1. *Blasting with Delay Caps*

(Use no zero delays. If zero delays are used, delay circuits must be treated as instantaneous circuits.)

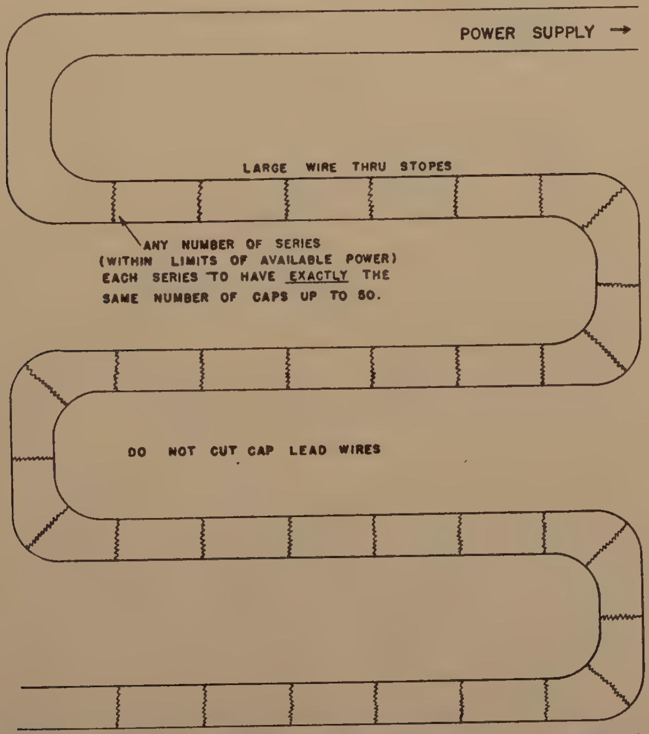
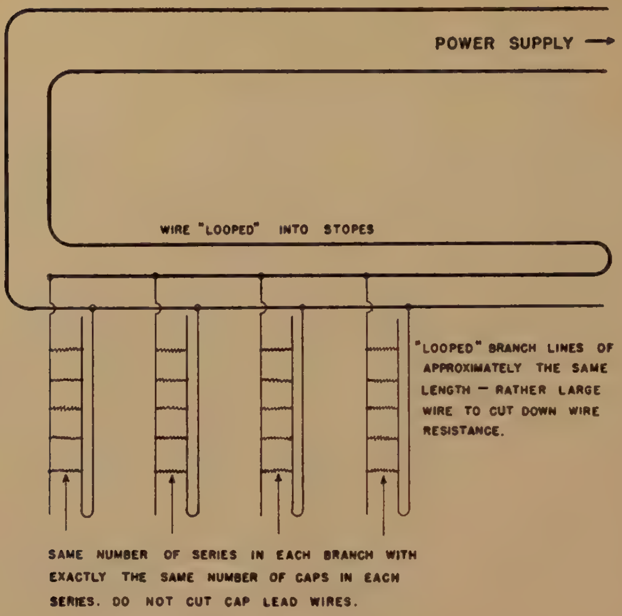

POWER SUPPLY →
LARGE WIRE THRU STOPES
ANY NUMBER OF SERIES (WITHIN LIMITS OF AVAILABLE POWER) EACH SERIES TO HAVE EXACTLY THE SAME NUMBER OF CAPS UP TO 50.
DO NOT CUT CAP LEAD WIRES

FIG. 8.—TWO-WIRE SYSTEM WITH LADDER CONNECTIONS BUT WITHOUT BRANCHES.

1. Since a balanced system is not required, use a two-wire system with branches and regular "ladder" hookup (Fig. 7), to reduce resistance and to simplify hookup.

2. Since any combination of cap series may be used, dummy caps are not needed, therefore a serious safety hazard is eliminated. Use no dummy caps. Cut cap lead wires to make snug connections.

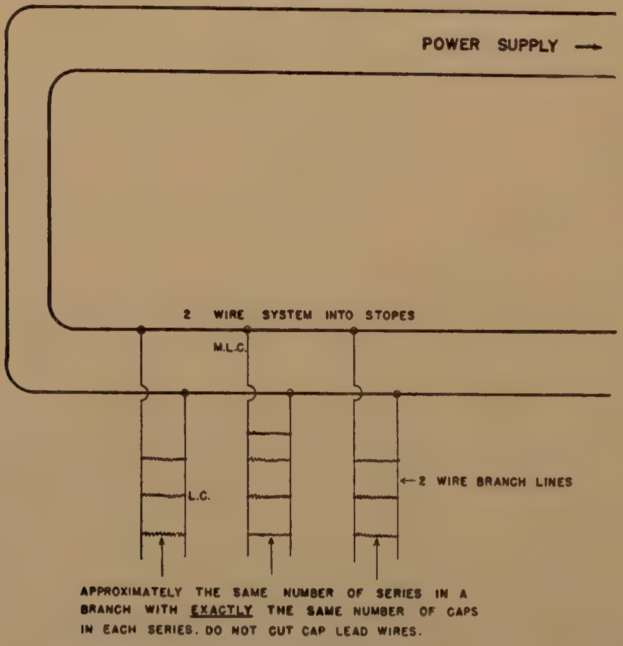


"LOOPED" BRANCH LINES OF APPROXIMATELY THE SAME LENGTH — RATHER LARGE WIRE TO CUT DOWN WIRE RESISTANCE.

SAME NUMBER OF SERIES IN EACH BRANCH WITH EXACTLY THE SAME NUMBER OF CAPS IN EACH SERIES. DO NOT CUT CAP LEAD WIRES.

(FOR USE IF BRANCHES ARE ADJOINING)

FIG. 9.—LOOP BACK. A THREE-WIRE SYSTEM WITH BRANCH LINES APPROXIMATELY SAME LENGTH.



APPROXIMATELY THE SAME NUMBER OF SERIES IN A BRANCH WITH EXACTLY THE SAME NUMBER OF CAPS IN EACH SERIES. DO NOT CUT CAP LEAD WIRES.

(FOR USE IF BRANCHES ARE NOT ADJOINING)

FIG. 10.—A TWO-WIRE SYSTEM.

3. Have ample factor of safety of resistance and current requirements.
4. Have blasting switch on secondary leads from transformers.
5. Use extreme care in making all electrical connections, both in stopes and main leads.

Case 2. Blasting with Instantaneous Caps

1. If the stope layout will permit a two-wire system, *without branches*, with "ladder" connections of up to 50 caps in series can be used (Fig. 8), use it, to cut down wire resistance and simplify hookup.
2. If the stope layout requires branches, and branches are adjoining, use "loop back" or three-wire system (Fig. 9). If branches are not adjoining, use two-wire system (Fig. 10).
3. Use *exactly* the same number of caps in each series. Do not cut cap lead wires.
4. Have ample factor of safety of resistance and current requirements.
5. Have blasting switch on secondary loads from transformers.
6. Use extreme care in making all electrical connections, both in stopes and main leads.

STRAY CURRENTS

In all shots fired electrically there is the hazard of stray currents, particularly lightning, causing premature explosion. In most large mines where power is used underground, leakage from various sources causes all metallic conductors, such as track and pipe lines, to become a source of danger, and precaution should be taken to avoid premature explosions from this source. Our procedure in blasting heading and raise rounds, as well as big blasts, include precautions against stray currents.

ACKNOWLEDGMENTS

The author wishes to thank Mr. J. R. Sweet, Mine Superintendent; members of the mine staff; representatives of General Electric Co., Hercules Powder Co. and DuPont Powder Co., for aid and criticism in preparing this article, and the Management of the Climax Molybdenum Co. for permission to publish it.

LITERATURE

1. W. J. Coulter and W. E. Romig: A Record Underground Blast. *Eng. and Min. Jnl.* (Oct. 1933). Description of a large blast at Climax.
2. F. S. McNicholas: Blasting Big Rounds Electrically. *Eng. and Min. Jnl.* (Oct. 1932). Detail of wiring, testing, connecting, calculations, units of labor.
3. R. L. Healy and F. S. McNicholas: Large Scale Blasting. *Canadian Min. Jnl.* (May 1935). General description of area and methods. Detail of calculations. Blasting from remote control. Costs.

DISCUSSION

(J. Murray Riddell presiding)

G. Sherman,* New York, N. Y.—As this undercutting blasting seems to resemble a continuous caving operation, I wonder why it seems to be desirable to fire so many blasts at once. It is customary in the porphyries to shoot fewer at a time and progressively cover the area.

F. S. McNicholas.—We have two reasons. One is that it has been customary at Climax to blast this way, and the other is that some of our ground is not really caving ground—we get tremendous stopes opened and they will not cave, which leaves a back that is open perhaps for 250 by 300 ft. When the adjoining stope comes up beside it we cannot blast pillars a few at a time and retreat because we are likely to get the entire back down, giving a dangerous air blast.

In caving grounds, where the ground would cave near the pillars that are taken out, we are using the other system of blasting a few pillars at a time, and it is preferable.

---

* Consulting Engineer.

Classification of Surface Mining Methods*

A special committee on the Classification of Surface Mining Methods was appointed by the A.I.M.E. Board of Directors on Nov. 17, 1939. The committee was headed by Gerald Sherman, and the members were Chester A. Fulton, C. F. Jackson, Earl E. Hunner and Arthur Notman. The following form of classification was reported to the Board by the Committee on May 16, 1940, as its unanimous recommendation.

SURFACE MINING

- Bank Mining
  - Bottom Benching
    - Dry Mining
    - Hydraulic Mining
  - Top Benching
    - Dry Mining
- Pit Mining
  - Bottom Benching
    - Dry Mining
    - Hydraulic Mining
    - Dredging
  - Top Benching
    - Dry Mining
    - Wet Mining
- Subaqueous Mining
  - River Bed
    - Top Benching
    - Dredging
  - Lake Bed or Ocean Bed
    - Dredging
- Placer Mining
  - Hand Mining
  - Ground Sluicing
  - Hydraulicking
  - Dredging

In most cases the terms used in the accompanying outline define themselves. Bank Mining is distinguished from Pit Mining by its situation. In Pit Mining the ore is taken from an operation below the level of the surrounding surface.

In the secondary classification, Bottom Benching is an operation in which a bench is attacked from below, but the material mined might be hauled up out of the pit or moved out horizontally or downward.

---

* Reprinted from *Mining and Metallurgy* (July 1940) **21,** 326.

In Top Benching the equipment is situated above the material to be handled, which is dragged upward for delivery at the point of disposal. A variety of mechanical devices for this purpose have been developed in the past few years, and enormous quantities of ore or waste are now moved by draglines, excavation cranes, etc.

In Hydraulic Mining, a stream of water under natural or pump pressure is directed against the bottom of a bank of earth or gravel, or soft rock, which is undermined, and the water, sand, or gravel is carried away in a sluice or flume.

Dredging would include all forms of dredges, clamshell, bucket-line, suction, etc.

Wet Mining may be practiced in a top bench by the use of an artificial pond above the natural water level to make it possible to use a machine floated on pontoons.

Placer Mining is classed as a specialty and the methods in common use are listed in a somewhat different form.

Controlling Subsidence of a Large Inverted Cone of Barren Rock Lying above the Ore Body, Colorada Mine, Cananea Consolidated Copper Company

By William Catron* and Cyril U. Cooledge,† Members A.I.M.E.

(New York Meeting, February, 1938)

Because the rich La Colorada orebody of the Cananea Consolidated Copper Co. does not outcrop at surface, after its discovery (by churn drill) and before mining was begun, a large amount of development work was done to learn the shape and extent of the ore and to provide for economic and effective mining. As these openings progressed, the actual shape of the orebody became known and with this knowledge came the realization that mining this rich and unusual deposit without loss of ore or danger to workmen presented several difficult problems. This paper will describe one of these problems; namely, the handling of an inverted cone-shaped core of barren rock, on the periphery of and below which the ore occurred. It was clear that as the ore was mined out from around and under this inverted cone of rock, weighing about 3,000,000 tons, it would hang over the workings, held suspended only by its own tensile strength and threatening to crash into the stopes below with disastrous results to men and mine.

The method of mining developed during the years 1928, 1929 and 1930 anticipated and provided for the eventual collapse of this core of rock. In November 1936 the core settled down gradually on the edges of the stopes, descending a total of about 50 ft. vertically. The men in the stopes had ample warning and no lives were endangered. There were no air blasts in the mine. The core broke loose from its top support at the 500-ft. level along old fracture planes and evidently is crushed to the ninth level. Movement was of short duration; the following shift went to work in the stopes as usual. No ore was lost.

Geological Structure

The shape of the orebody is illustrated in Fig. 1. The geology of the Colorada orebody has been described by V. D. Perry[1] and by W. G.

Manuscript received at the office of the Institute Dec. 11, 1937. Issued as T.P. 938 in Mining Technology, July 1938.

* Superintendent of Mines, Cananea Consolidated Copper Co. S. A., Cananea, Son., Mexico.

† Chief Engineer, Cananea Consolidated Copper Co. S. A.

[1] V. D. Perry: Copper Deposits of the Cananea District, Sonora, Mexico. Copper Resources of the World, **1**, 413–418; *Bull.* Geol. Soc. Amer., **47**, 53–86.

Valentine.[2] The geological structure, which had an important bearing on the problem, is briefly as follows:

At La Colorada a quartz porphyry intrusive cuts through fine-grained volcanics. The orebody is found in a pipelike structure with an elliptical plan section within the quartz porphyry and at the contact of the porphyry and the volcanic rocks. Below the tenth level glassy quartz and high-grade copper sulphide mineralization extend across the entire structure; above that level the mineralization follows an expanding ring-shaped fracture, which bounds a core of waste rock in the form of an inverted cone. The ore above the tenth level consists of quartz and

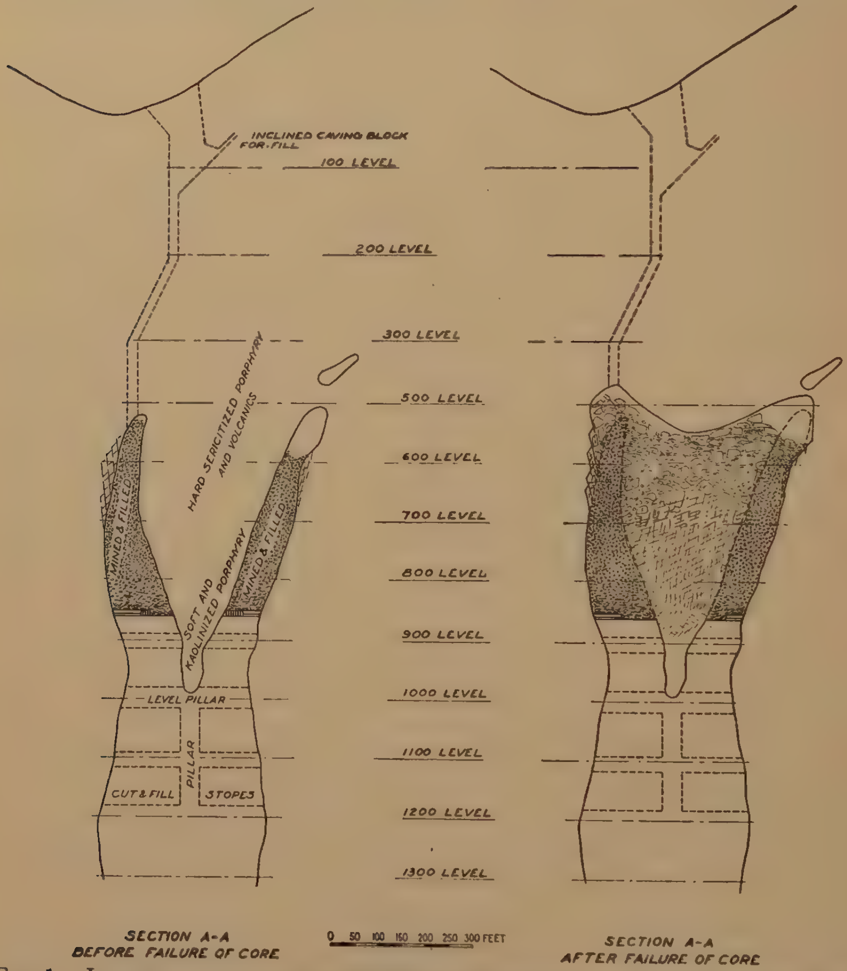

INCLINED CAVING BLOCK FOR FILL
100 LEVEL
200 LEVEL
300 LEVEL
500 LEVEL
600 LEVEL
700 LEVEL
800 LEVEL
900 LEVEL
1000 LEVEL
1100 LEVEL
1200 LEVEL
1300 LEVEL
HARD SERICITIZED PORPHYRY AND VOLCANICS
MINED & FILLED
SOFT AND KAOLINIZED PORPHYRY
LEVEL PILLAR
PILLAR
CUT & FILL
STOPES
0 50 100 150 200 250 300 FEET
SECTION A-A BEFORE FAILURE OF CORE
SECTION A-A AFTER FAILURE OF CORE

FIG. 1.—IDENTICAL SECTIONS THROUGH ORE BODY BEFORE AND AFTER CORE FAILURE.

---

[2] W. G. Valentine: Geology of Cananea Mountains, Sonora, Mexico.

copper sulphides localized within the ring-shaped fracture and as a network of seams and stringers in the wall immediately adjacent to the controlling structure. The ore has a subsurface apex just above the fifth level. At surface narrow quartz veinlets and iron oxide veinlets, which outcrop in a zone of 100 to 200 ft. wide over the north side of the structure, form the only surface expression of the high-grade orebody. Extending upward from the ninth level a short segment of the northeast part of the ring is not commercially mineralized. The lower portion of the waste core has been weakened by intense alteration and post-mineral brecciation of part of the core and the ore minerals has been developed around the interior of the ore ring. This brecciation extends irregularly upward from the tenth level around the inner perimeter of the ring, and on the southeast side reaches the surface, 800 ft. above the fifth level.

The lower end of the core has been thoroughly kaolinized and is structurally weak, while above the 700-ft. level the core, although fractured, has been cemented with quartz and sulphides and shows little kaolinization. The joints that appear on the outside of the orebody are weak or absent within the core. The controlling factor of caving within the core will be these fracture planes.

MINING

The orebody was developed to be mined as follows: A vertical pillar 40 ft. wide was laid out directly under the long axis of the core. On either side, normal to this axis and extending vertically to the top and bottom of the orebody, stopes 30 ft. wide separated by vertical pillars 50 ft. wide were to be mined by horizontal cut and fill. After cut-and-fill stoping had sufficiently advanced, top slicing was to begin at the top of the 50-ft. pillars. The level interval was 125 ft. and to protect the levels and give lateral strength to the 50-ft. vertical pillars, horizontal floor pillars 20 ft. thick were to be left standing over the cut and fill stopes, and pillars 11 and 22 ft. thick were left standing over the levels underneath the cut and fill stopes. Above haulage levels a pillar of 22 ft. was left; over intermediate levels an 11-ft. pillar was left. As top slicing descended to the elevation of these horizontal pillars, they were to be mined by reaching out from the top-slice floor.

The character of gob to be used in the horizontal cut-and-fill stopes was given careful consideration. A quarry in kaolinized tuff was opened near the portal of the mine and the material hauled to a storage raise in the mine. This material had certain very interesting and important qualities. It spread easily by scraper in the stopes and disintegrated quickly when so handled. When sprayed (to allay dust) or when wetted by waste water from machine drills, it packed tightly. This natural tendency to pack was helped by blasting down the horizontal slices of ore on top of each layer of fill, after it was leveled off and planked. It was desirable to have the cut-and-fill stopes left tightly packed with fill in order to prevent sloughing of the sides of the 50-ft. vertical pillars, and to avoid the necessity of gob fences as top-slice stopes descended. As anticipated, a great advantage was realized from the use of this particular kind of gob, and the manner in which it was packed. It was found that this gob had packed too tightly in the cut-and-fill stopes, that when top-slice miners broke to the gob it stood vertical for the full height of the slice, over a considerable length and for a long period of time. No lacing was necessary in the top slices to hold it back.

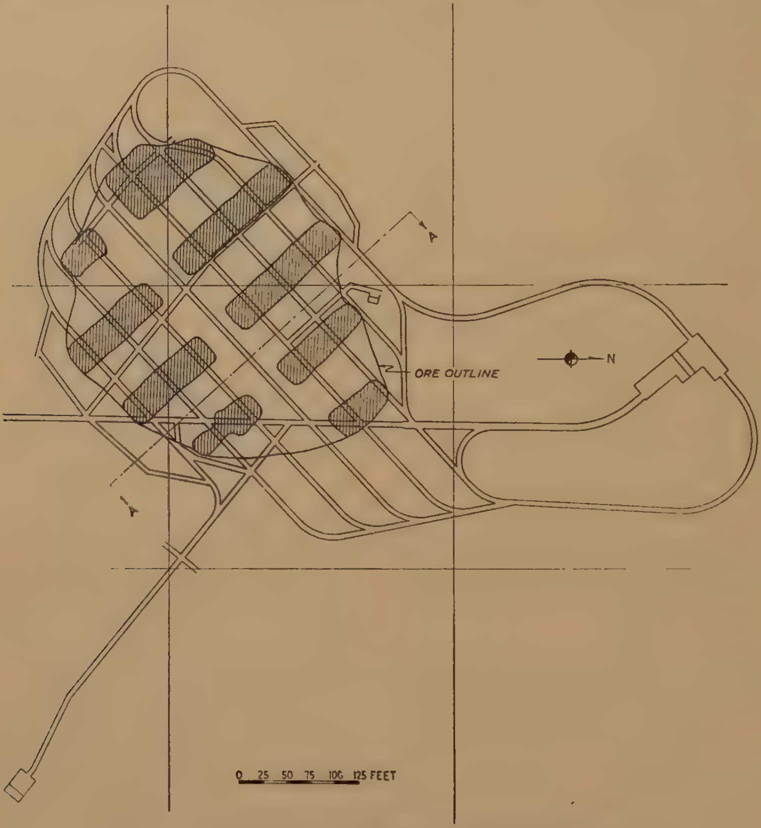


FIG. 2.—CUT-AND-FILL STOPES ON 1000-FOOT LEVEL.

It was also believed that the tightly packed fill between the pillars would have another beneficial effect. When the core broke loose and crushed downward, stresses would be transmitted laterally into the pillars just below the top-slice floors. Because of the tightly packed filling between these pillars, the crushing effect of this transmitted stress, it was thought, would be limited in extent. Actually, when crushing took place, it was found to be localized near the edge of the pillar near the core.

In addition to providing gob for cut-and-fill stopes, it was decided to fill on top of the empty space that would be created above as the top-slice stopes descended. A shrinkage stope about 250 ft. long was started from the 500-ft. level on the west side of the ore and carried to surface. The lower part of this stope was in low-grade ore, the remainder was waste and was used for fill. The pillar on the 500-ft. level (under the shrinkage stope) was removed after top slicing began, and the waste ran directly into the stoped-out space below. Where this shrinkage stope broke through to surface, quarrying operations were undertaken to provide filling material at low cost. Waste is also hauled and dumped on top of the mined area not covered by the waste that falls from the quarry above the west side of the orebody. These provisions for keeping the spaces above the stopes filled with waste served to cushion the effect of the caving of the core and at the same time guarded against the possibility of an air blast, should the core fall suddenly.

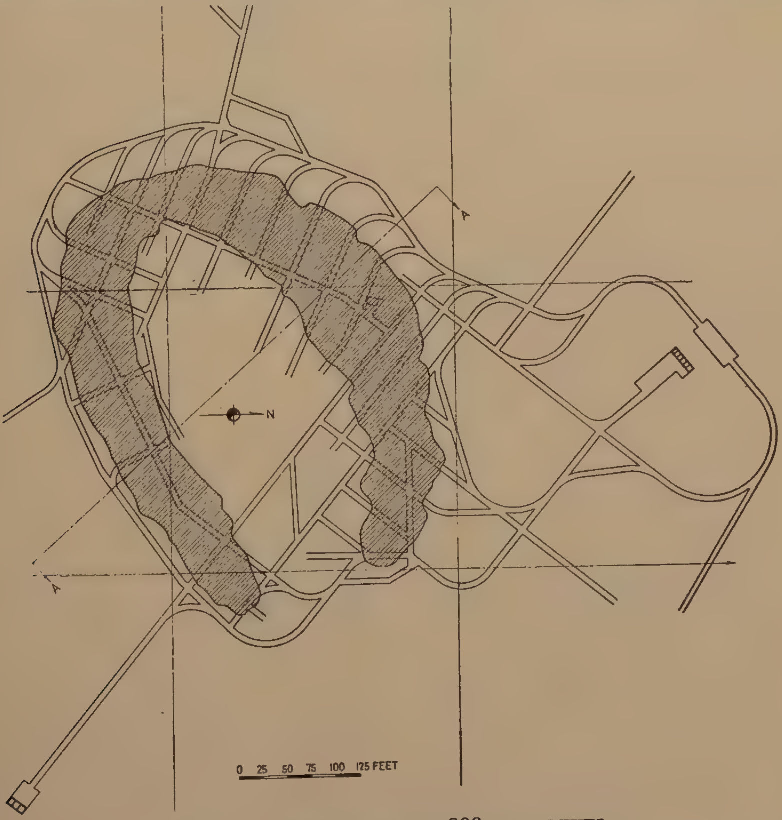


FIG. 3.—MINED AREA ON 800-FOOT LEVEL.

Top slicing had nearly reached the 800-ft. level before any signs of failure of the core were noticed At this time a break appeared on the

600-ft. level in 6-25 and 6-11 crosscuts. Movement along these breaks was gradual as the ore below was removed and the waste filling subsided. No damage was done to the stopes below. Three years later, when the top-slice stopes were about halfway between the 800 and 900-ft. levels, another movement in the core took place. Cracks opened on the 500-ft. level and on the 600-ft. level the line of break in 6-11 crosscut moved back into the core about 80 ft. This movement was gradual, cushioned on the fill above the stopes, and did no damage to the stopes below.

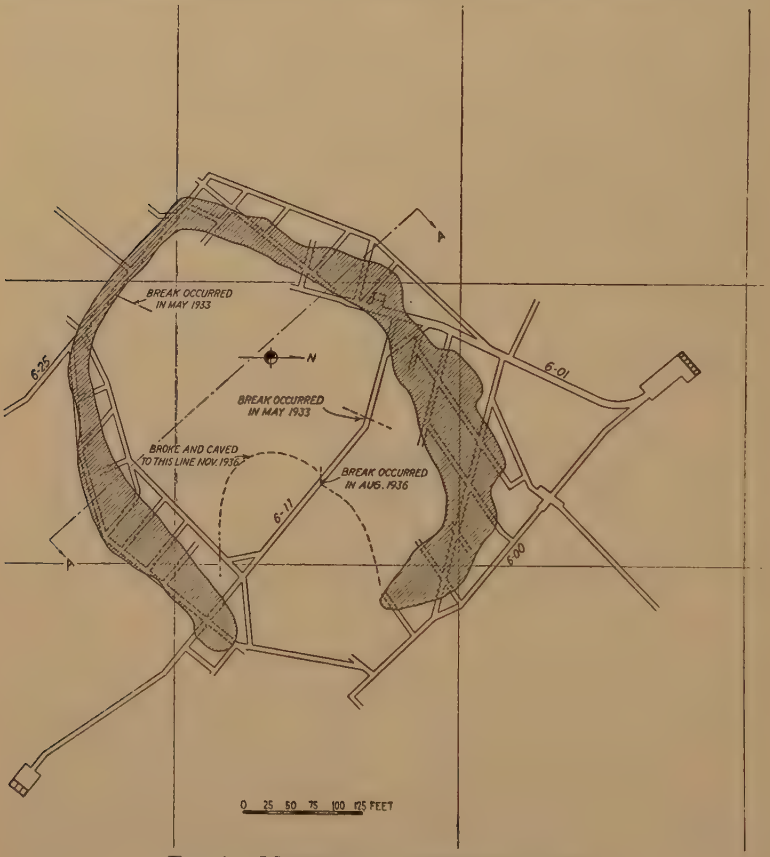

BREAK OCCURRED IN MAY 1933
6-25
N
A
BREAK OCCURRED IN MAY 1933
6-01
BROKE AND CAVED TO THIS LINE NOV. 1936
BREAK OCCURRED IN AUG. 1936
6-11
A
6-00
0 25 50 75 100 125 FEET

FIG. 4.—MINED AREA ON 600-FOOT LEVEL.

In November 1936, when the top-slice stopes were in places near the 900-ft. level, the core suddenly failed. The immediate effect was the gradual crushing of the open top-slice stopes and the level timbering under those stopes near the level. The tops of four untimbered raises between the 1000 and 900-ft. levels in the soft kaolinized ore near the core were broken and caved. Otherwise no damage was done below the 900-ft. level. The men in the stopes had ample warning and no lives were endangered. There were no air blasts in the mine. The core broke loose from its top support at the 500-ft. level along old fracture planes and evidently is crushed to the 900-ft. level.

As soon as the movement ceased, top slicing was resumed and in a few days was back to normal production. There was a drop in production during the ensuing three months due to mining through the level and having to remove the ore by square sets under places where the stopes caved before the floor was laid.

The cause of the final failure was probably the previous weakening of the core by breaking along old fracture planes and movement along a new break with excessive weight thrown on the structurally weak kaolinized porphyry at the base of the core.

There is evidence that the core at bottom is completely crushed and that the upper part is in big blocks, but not so big as to concentrate weight on any one spot.

The filling material between core and outside boundary of orebody is not compressed too tightly, but shrinks down on top as the top slices descend, probably because of the width of the filled orebody. Had the orebody been narrow, the fill might have compressed so tightly as to hang up. As a precaution, as this fill shrinks down, more waste is quarried on surface and the space kept reasonably full.

Some Outstanding Mine-hoisting Equipment

By Bruno V. E. Nordberg*

(New York Meeting, February 1940)

Hoisting is one of the earliest endeavors of man with machinery, for hoisting was probably used by the early Egyptians. Treadmills were used for general hoisting until early in the nineteenth century and it was a short step only to attach a Watt steam engine to the rope drum of a hoist. One of the earliest steam hoists was built for a coal mine at Frisne, near Valenciennes, France. It was a remarkable piece of engineering created by James Watt in 1811, using a wooden drum on a vertical shaft. This followed the crude hoist by Holtzhausen in 1803, which used a drum of wood on a horizontal shaft.[1] In this country there were examples of crude mine hoists, which have been developed into the efficient machines in use today. African and Australian mines show examples of both extremes, and have an array of fine hoisting equipment exemplified in those built by British manufacturers. The influence of both English and American engineering in hoisting is interestingly evident in Canada. The information available to the writer does not reveal this to be true of South Africa, although American-built hoists have been furnished.

Mining camps develop their own style of equipment, adapted to local conditions. For instance, the Butte mining district adheres to the straight drum with friction clutches, because of the great number of levels from which hoisting is done or to which men and material are lowered and the necessity for hoisting or lowering unbalanced. The north Michigan copper country uses both straight and cylindroconical drums and hoists continually in balance, so that the drums are not clutched to the main driving shaft. The Joplin mining district, Missouri, has its own peculiar small hoists. Other individual mines or localities in which a number of mines have been developed show individualistic types of hoists, depending upon their particular problems. In each instance, however, the particular problem must be carefully analyzed, as conditions are seldom so near the same that a number of identical hoists can be used to advantage.

Manuscript received at the office of the Institute Dec. 1, 1939. Issued as T.P. 1190 in Mining Technology, September 1940.

* Executive Engineer, Nordberg Manufacturing Co., Milwaukee, Wisconsin.

[1] References are at the end of the paper.

Development has generally passed from the use of steam as a motive power to electric drive, owing to the enormous expansion of electro-generating plants and the extension of high-tension power lines. An air hoisting system is used in the Butte district, Montana,[2] and the use of steam is continued in the North Michigan copper country. Coal mines show a large variety of equipment, both large and small, steam or electric drive, and generalization is not possible. The most highly developed as well as the crudest forms of hoisting equipment can be found in coal fields.

Choice of Hoist

A hoist may be defined as a drum arranged for winding or propelling rope, the drum being driven by whatever energy is most economically available and controllable from a single location or from two or more positions. A hoist drum may be driven directly by an electric motor, steam or air engine, either direct-connected or geared, and controllable as to speed and output of energy. Other forms of drives through friction clutches, belts or other means, from a source that is in continuous operation, are common generally to small hoists, but may be used to handle even very large hoists. On the property of the Calumet and Hecla Cons. Copper Co., at Calumet, Mich., a 2000-hp. reciprocating steam engine drives a long shaft to which six hoist drums, each 25-ft. dia. by 8-ft. face, are geared and driven through band friction clutches, each hoisting, unbalanced, a cage in a different shaft and each having a separate operator to handle the particular drum through signals in the respective shaft.

If a hoist is required to pull a great tonnage in a very short time, heavy peaks in power consumption result and the cost of electrical equipment to hold down these peaks may militate against its adoption. To prevent imposing heavy peaks on the power line, the well-known Ward Leonard control has been adopted, the short peaks being taken care of by the energy in the flywheel. In England and South Africa, some outstanding installations use a condensing turbine driving a flywheel set, there being no large electrical system from which to draw. The relative merits of this system over that of driving the hoist drum directly by steam is not clear, but is described in literature.[3]

Coal or other fuel prices and the availability of water will influence consideration of a steam plant direct-connected to the drum of a hoist, or the use of electrical energy, or other type of plant. The Burro Mountain Copper Co., when operating at Tyrone, N. M., used Diesel engines to generate electrical energy for hoisting, pumping and general power purposes, not including the compressing of air. Air compressors are separately driven by being direct-connected to Diesel engines.[4] Electrical energy transmitted long distances to compress air for use in a large pneumatic hoisting system was amply discussed in a paper in 1913.[2]

SPECIAL INSTALLATIONS

The installation at the Inspiration shaft of the Inspiration Copper Co., Inspiration, Ariz., using an entirely automatic system of hoisting from a single level, contains two hoists, electrically driven, operating consecutively in order to use the inertia of the hoist, load and rope of the moving hoist in overcoming the momentum during acceleration in starting the other hoist. In this manner, a saving in power necessary for acceleration is attained.[5] It is a successful installation handling 20,000 tons of ore a day. It is an example of a specific problem, which required a specific solution.

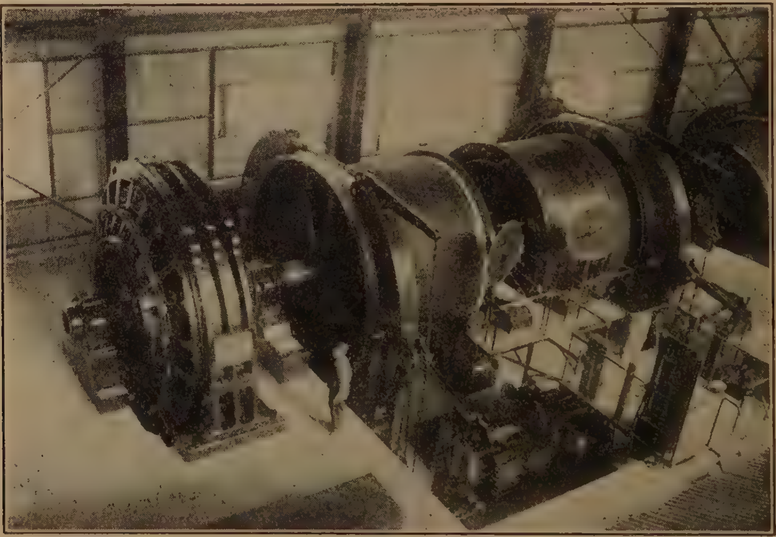


FIG. 1.—ONE OF TWO AUTOMATICALLY OPERATED HOISTS AT INSPIRATION COPPER COMPANY.

Double 10-ft. dia. drums; d.c. motor, 580 hp., using common flywheel set. One hoist started as other is stopped. Built by General Electric Co. and Nordberg Manufacturing Co.

Another automatically operated hoisting plant is that of the Mufulira Copper Mines Ltd. at its Selkirk shaft, Northern Rhodesia. Two hoists alternately lift skips 1000 ft. (ultimately 2600 ft.), each skip containing 22,000 lb. of rock. Each double-drum hoist is driven by a 1600-hp. motor through Bibby couplings. The two hoists have two motor-generator sets, each being a 1120-kw. direct-current generator driven by a 1200-hp. synchronous alternating-current motor at 750 r.p.m., with flywheel weighing 11 long tons.[6] Each hoist lifts 11,100 tons of rock per day. Other examples of automatic hoisting can be found in Australia, South Africa and Great Britain.[7]

Conditions at the Orient No. 2 mine of the Chicago, Wilmington and Franklin Coal Co., West Frankfort, Ill., required the development of a hoist to handle from 1500 to 2000 tons of coal per hour from a depth of 607 ft. through the narrow limits of a single shaft. It required a single-drum first-motion hoist, driven by two 2200-hp. direct-current motors and called for 3¼ trips per minute. This installation has also been described[8,18] and need not occupy us further here, except to indicate that this problem made it necessary to design a special drum and to consume large blocks of power.

The handling of men and material has been solved in various ways, the solution being dependent not only upon local conditions but upon the installation of surrounding equipment. For instance, at one of the shafts on the property of the Phelps Dodge Corporation, an ordinary electric elevator cage is used, which is handled entirely from controls in the cage. In most cases, a straight drum hoist handled by an operator on the surface is a satisfactory means to meet this demand. Sometimes these hoists are large enough to handle rock in small quantities in cars in a cage. The man hoist built for the Homestake Mining Co. is of the same construction as the ore hoist and includes large cylindroconical drums, the large diameter being 25 ft. and the small diameter 12 ft., handling a rope of 1⅝-in. dia., and driven by one 1500-hp. motor. The cage of 10,000 lb. is counterbalanced by a weight of 18,000 lb. These few examples indicate a large variety of hoists used for this purpose which permit no generalization.

Unusual requirements have demanded a hoist that operates at high speed to handle ore and at slow speed to handle men, direct-connected to two motors of different sizes, as well as an additional steam engine, each separately clutched to the drum shaft. An example is the cylindroconical drum hoist, of the Chicago, Wilmington and Franklin Coal Co. at West Frankfort, Illinois.[8]

DRUMS

The type of hoisting drum is determined primarily by the amount of rock to be hoisted per unit of time. Secondary considerations are the cost of the equipment, kind of power to be employed and depth of mine. Straight drums are used extensively where the power consumption and hoisting speeds are low and where the first cost must be as low as possible; where a number of layers of rope are to be wound or where the depth of the shaft is indeterminate, as in exploration and development work. As long as rope speeds and acceleration are kept low, the straight drum hoist usually will answer the purpose. When hoisting from various levels, double drums are employed in which one or both drums are clutched. A satisfactory condition of balanced hoisting can be obtained for all levels. This type of hoist can be found in almost all mining camps. An example is shown in Fig. 2.

For hoisting from various levels, clutched drums are frequently used, arranged so that one or both may be clutched to the hoist shaft, but hoisting from several levels with a single drum is not uncommon, especially in such places as the north Michigan copper mines, where great depth makes the large cylindroconical drums necessary. It is clear that two movements of the drum are required for a single lift, except from one predetermined lower level, but where depths are great and long runs follow each double move, the losses are compensated for by the extended runs that follow. For instance, at the deep mine of the Quincy Mining Co. in northern Michigan the drum has been designed to wind 10,000 ft. of rope. The total trip is made in 3.3 min., with a rope speed of 3000 ft. per min. The power consumption during acceleration is small when compared with the total power consumption, in contrast with the hoist at the Chicago, Wilmington and Franklin mine, where the total running time is 16 sec., the acceleration is 14 ft. per sec. per sec. and the time of acceleration is 4½ sec. It is apparent that the time of acceleration of 4½ sec. compared with 16 sec. would not make it economical to hoist from intermediate levels, whereas drums clutched to the hoist shaft make possible hoisting from intermediate levels and bring about economy of power that is recognized generally. A unique arrangement was used by the Franklin Mining Co., at Hancock, Mich., when that company was operating. The hoist was driven by a steam engine having air cylinders

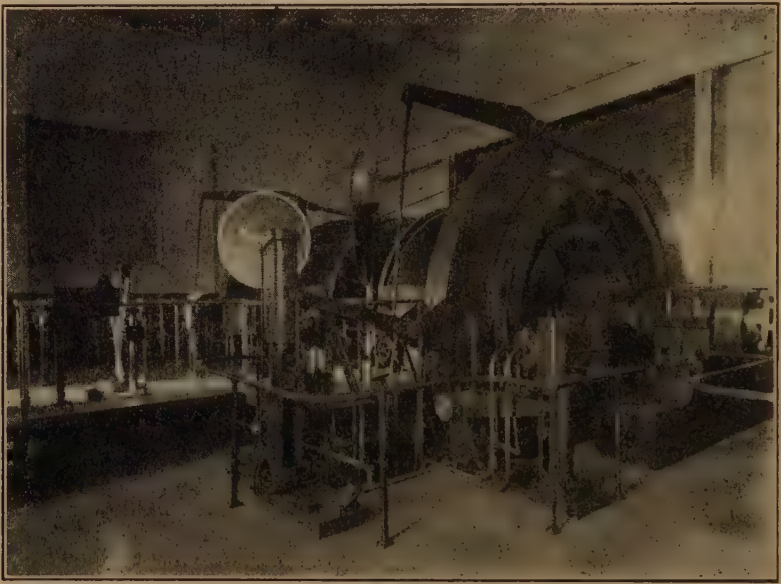


FIG. 2.—STRAIGHT-DRUM HOIST, BUILT FOR NORANDA MINES LIMITED. Drums 12 ft. by 78 in.; 9-ft. tooth clutch. D.c. motor, 2000 hp. Composite construction. Rope pull, 40,600 lb. Built by Canadian General Electric and John Bertram & Sons Ltd.

behind and in tandem, the purpose being to store energy during lowering by compressing air into the steam-generating system. A special valve gear made these cylinders ineffective during hoisting. When hoisting, the mixture of steam and air was used in the steam cylinders, enabling the return of energy sufficient to approximate the gain in balanced hoisting. The statement has been made that the more expensive hoisting equipment was amply paid for by the savings in sinking a single instead of a double shaft.

ROPE WEIGHT

One of the best means of balancing the rope weight on a hoist is with reels. This balance is maintained over the entire length of lift by properly selecting the rope and reel diameter. Hoisting can be done from any level with a minimum power consumption. The WR^2 of the drum becomes very small, so that the power required for acceleration is low. The disadvantage lies in the need of a flat rope, usually made by lacing together several ropes of small diameter. The first cost and expense of maintenance of such ropes is high; the first cost of the hoist, however, is fairly low. The reel may be clutched to the hoist shaft, so as to suit hoisting conditions from various levels. Much use has been made of reel hoists in France (Fig. 3) and several can be found in this country.

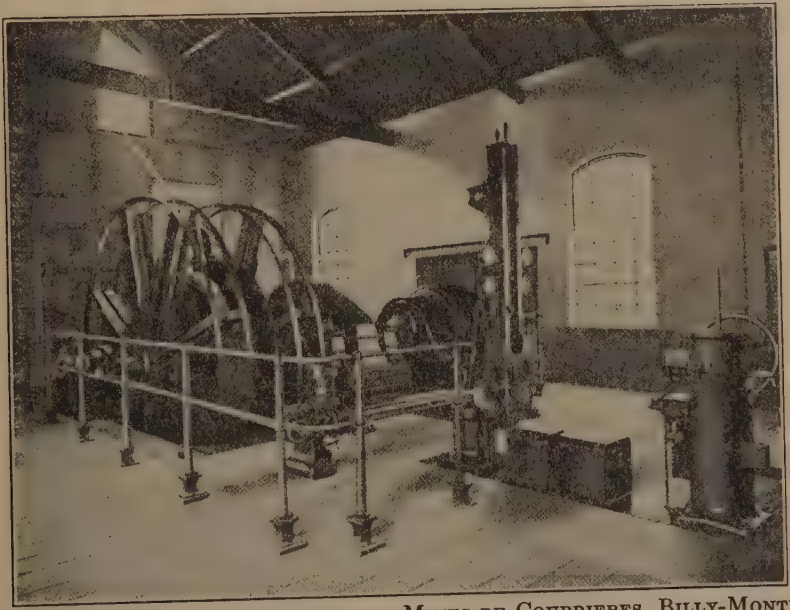


FIG. 3.—REEL HOIST AT COMPAGNIE DES MINES DE COURRIERES, BILLY-MONTIGNY (PAS-DE-CALAIS) FRANCE.

Motor, 430 hp., 50-cycle, three-phase. Forty built for reconstruction program following war. Built by Maison L. P. Durand, Chalon-sur-Saône (formerly Établissements Galland).

Attempts have been made to completely balance the loads and ropes or to partly balance these to the best advantage of power consumption. Such attempts have resulted in special drum forms, like the conical drum, cylindroconical with a medium slope and cylindroconical with a steep slope, the latter being known as the spiral-groove drum. These can be built either in the single or double drum, side by side or one behind the other, or in tandem. A single cage at one end with a counterweight at the other end has been used with a cylindroconical drum by the Morton Salt Co., at Grand Saline, Tex. An unequal cylindrical portion of the small diameter of the drum causes overbalancing of the counterweight and aids in propelling it upward. The cylindroconical drum with a spiral groove is designed to aid a loaded skip upward and to reduce the power needed for acceleration. To accomplish this, the large drum diameter may be chosen so that the weight of the empty skip overbalances the loaded skip and rope at the loading station, by hoisting the loaded skip on a sufficiently small diameter of the drum. During acceleration of the drum and as soon as a predetermined rotative speed has been attained, the rope is "shot" from the small diameter up the spiral groove to the large diameter, thereby increasing the speed of hoisting to the maximum. This type of drum is of great benefit in power saving, and is designed for hoisting from a single level only. It is however, used for hoisting from various levels, as, for instance, in South African mines (Fig. 4).

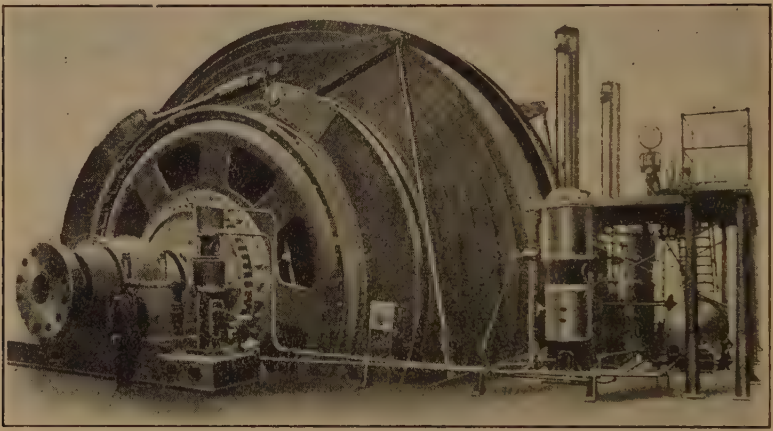


FIG. 4.—DIRECT-DRIVEN ELECTRIC HOIST WITH DOUBLE CYLINDROCONICAL DRUM. MADE FOR NO. 4 SHAFT, CITY DEEP LTD., SOUTH AFRICA.

Drums 17 ft. 6 in. to 35 ft. in diameter by 12 ft. ¾ in. wide over-all. Length of wind, 4500 ft. Built by Fullerton Hodgart and Barclay Ltd., Vulcan Works, Paisley, Scotland.

CLUTCHES

Many types of clutches have been offered, the earliest being a dog or toothed clutch. Where cages and cars are employed, the toothed clutch, particularly if its diameter is too small, does not permit the setting of the drums to the desired absolute level difference necessary and at the end of each trip may require maneuvering of the hoist. With skip hoists too this disadvantage obtains. Consequently a development of the friction clutch followed. One type consisted of a metal strip or band gripping a clutch drum. The crudeness of the mechanism for operating these band clutches led to some disasters in the South African mining field. A disk friction clutch gradually supplanted the band type in this country. The relative merits of the friction and toothed clutches are still debated in the Canadian fields and the toothed clutch is almost entirely used where the hoists are supplied by British manufacturers. The ease with which a friction clutch can be engaged and with which the relative positions of the lower and upper skip can be secured, argues much in its favor. It must necessarily be properly constructed to act efficiently. The toothed clutch requires accurately "spotting" the clutch member and the drum. Both arrangements require the same mechanism for engagement. There is no force in the argument that a large drum can be secured more safely with the toothed clutch than with the friction clutch, as is manifest in Fig. 5.

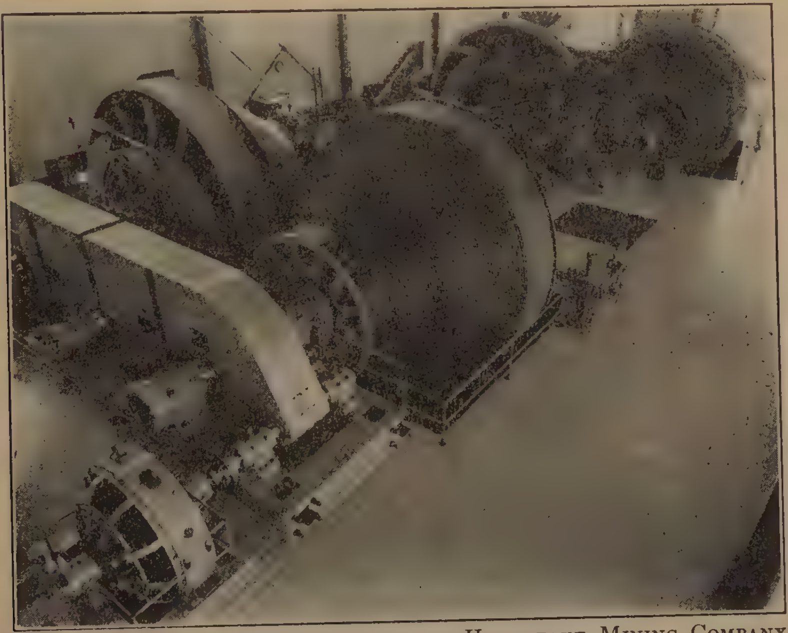


FIG. 5.—ORE AND MAN HOISTS BUILT FOR HOMESTAKE MINING COMPANY. Cylindroconical drums 25 and 12-ft. diameters; clutched. Two motors on ore hoist (foreground) and one on man hoist. Built by Nordberg Manufacturing Co. and General Electric Co.

The hoists built for the mine of the New Consolidated Gold Field, Ltd., of South Africa, use a toothed clutch (Fig. 6) and are of sufficient interest to be described in some detail. Both drums of the hoist are clutched, with an 8-ft. 4-in. toothed clutch having 60 teeth, these drums having a large diameter of 35 ft. and a small diameter of 13 ft., among the largest yet built, winding a rope for a vertical depth of 6660 ft. Three such hoists are at the Simmer and Jack Gold Mines Ltd. and two at the Sub-Nigel Limited Mines. The three hoists at the Simmer and Jack mines are placed in one hoist house, with ropes leading to one large headframe erected over a shaft where six skips, 9000 lb. each, are handled. Ward-Leonard control is used in conjunction with the flywheel set, which is driven by a steam turbine. The hoist makes 21.6 trips per hour from a depth of 6660 ft., drum revolutions being 65.3, rope speed 3000 ft. per min., rope diameter 2 in. and weight 6.89 lb. per foot; factor of safety 4.8, with a load of 16,000 lb. The total static and dynamic torque is 1,000,000 ft-lb. Acceleration to maximum speed of 27.3 r.p.m. occurs in 17.6 sec. with the rope on small diameter. The skip, rope and load receive a second acceleration during constant speed of the drum when the rope travels from the small to the large diameter, this second acceleration occurring in 17.6 sec. also. The total trip is scheduled for 2 min. 41.2 sec., with 5-sec. loading and dumping periods.[3]

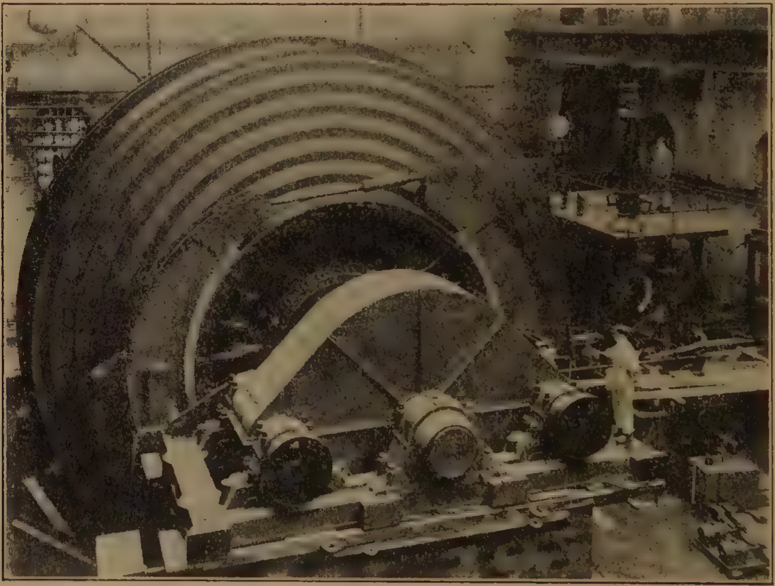


FIG. 6.—TOOTHED CLUTCH 8 FEET 4 INCHES IN DIAMETER ON GEAR-DRIVE, CONICAL-DRUM ELECTRIC HOIST. ONE MADE FOR SIMMER AND JACK MINES, LTD., SOUTH AFRICA.

Drums (2 loose), 13 to 35 ft. Suspended load, 61,200 lb. Length of wind, 6,300 ft. vertical. Built by Fullerton Hodgart and Barclay, Ltd., Vulcan Works, Paisley, Scotland.

Clutches of the friction type, 14-ft. normal diameter, are employed on the two hoists now installed and two hoists being built for the Homestake Mining Co. at Lead, S. D. One hoist handles men and is driven by one motor; the other handles ore, is driven by two motors, two cylindroconical drums in tandem, 25-ft. large diameter and 12-ft. small diameter, winding 5400 ft. of 1⅞-in. dia. rope, weighing 5.63 lb. per ft. Factor of safety is 4.3. The maximum rope speed is 2750 ft. per min. with a maximum drum speed of 35 r.p.m. The skip weighs 12,500 lb. and the load is 14,000 lb. The two motors, 1500-hp. direct-current each, with 3000 maximum horsepower, are gear driven, the two conical drums in tandem. Each clutch is capable of holding 325,000 ft-lb. The total time scheduled for hoisting is 155 sec., with 5-sec. rests.

Some Canadian mines have shown preference for the toothed clutch on large hoists and Canadian manufacturers have developed several types of these, as described in a contemporary paper.[9] The opinion expressed in that paper indicates that friction clutches are adaptable only to small hoists. However, the hoist being built for the Hudson Bay Mining and Smelting Co., to be installed at Flin Flon, Manitoba, indicates that this feeling is not general. An ore hoist is being built for that company, having two conical drums in tandem, 25-ft. large and 12-ft. small diameter, to wind 4634 ft. of 2-in. rope weighing 6.4 lb. per ft. The factor of safety is 5.0. Rope pull is 66,780 lb., while the 16-ft. friction clutches are capable of handling 575,000 ft-lb. torque. These clutches are designed so that torque may be increased to 850,000 ft-lb. This hoist is almost the same as those shown in Fig. 5.

Two hoists at the Dome Mines Limited, South Porcupine, Ont., Canada, have two straight 10-ft. drums, winding in three layers, both tooth-clutched to the drum shaft by laterally moving the toothed element. They hoist 3200 ft. with a maximum speed of 1400 ft. per min.[9,10] Clutching must be done by moving the toothed element laterally and rotating the shaft until the element slips into mesh. Vertical spacing of the skips can be only approximated by this method.

Another interesting hoist is one on the Johnson Company's property, Thetford mine, Quebec, Canada. It has an 8-ft. dia. drum driven by a 450-hp. motor and hoists a car weighing 10 tons with a 20-ton load of rock up a 22° slope 1000 ft. long at a rope speed of 700 ft. per min. The hoist is controlled near the dump, to enable the operator to see the car and watch its operation. The friction clutch is of much the same construction as that built for the Hudson Bay hoist described in an earlier paragraph.

BRAKES

As far as details are concerned, there is a general sameness about brakes for the drum. A rather ugly, but inexpensive, construction is displayed in the nutcracker brake on the hoist at Brakpan mines, South Africa. The influence of this construction has reached from South Africa to this country. A similar construction is that of an American-built hoist, and the most that can be said of this type of brake is that at the time it was constructed it was considered less expensive than the cast-iron brake shoe in vogue. For large installations, a large braking force is necessary and the accomplishment of this is most readily achieved by a large multiplication of movement. It follows that advantage can be gained most readily by reducing the movement of the brake shoe to a minimum. This calls for accurate parallel movement of the shoes equally from each side of the brake drum and leads to the form of hookup developed in this country many years ago.[11]

Faulty and improper arrangement of controls of hoist brakes led to a series of interesting developments of tempered or governed braking. A device made in England was designed to prevent excessive application of the brake, as well as sudden and unnecessarily severe braking impulses, by controlling the application of the brake in such a manner that the rate of retardation is limited to a predetermined value.[12] This makes possible substantial reduction in rope stresses and particularly prevents repeated rope stresses during braking. Smooth braking is important and can be accomplished by precise control of the brake by the operator, if the equipment is adequate. When hoisting is interfered with by one of many causes and a safety stop takes charge, braking should be controlled to prevent shock and severe stress on the rope and hoist parts. Such control is accomplished by step application; that is, increasing the braking application as the hoist speed decreases.[13] Means have been contrived to make resistance to the movement of the brake lever in the operator's hands apparent as the brake pressure is increased. Complications of the hoist should be avoided and sound construction of the controls makes such complication of questionable value.

Pulleys

Unusual hoisting equipment leads to consideration of a special friction drive. Fig. 7 shows the underground installation at the Park Utah Consolidated Mining Co., near Park City, Utah, where an 8-ft. Bollen pulley handling a 12,000-lb. load in a 5000-lb. skip hoists up 1000 ft. at 800 ft. per min., on a 1¼-in. rope. The hoist is driven through two sets of gears by a 250-hp. direct-current motor. The load is counterbalanced by use of a counterweight of 10,000 lb. The hoist stands directly over the shaft. The mode of increasing the tractive effort is shown in Fig. 8. As the pressure of the rope acting toward the center of the pulley on the central ring seat causes the ring to be depressed, it draws the jaws of the pulley proper together, and thereby the rings *B*, which clamp the rope. One member of the pulley is free to rock on a spherical bearing *D*, making possible the action described (Fig. 9). As one side is clamped tight, the opposite side opens to permit the rope to leave or to enter between the rings *B*. In this type of pulley the rope creeps to some extent, but this seems to have little or no effect on rope wear. The Bollen pulley is attractive as an underground installation, because of the small space necessary to house it. A half wrap of rope or even 150° contact produces the necessary tractive effort. This pulley is not designed for lowering large weights, unless a counterweight or a counterpull on the rope is provided.

An installation of a Bollen pulley at Superior, Ariz., for the Magma Copper Co., handles one skip and a cage in balance with a three-deck cage and produces a counterweight totaling 11,500 lb. on a rope of 1¼-in. dia. and using a tail rope of the same diameter. The total lift is 4000 ft.; load, 8000 lb.; rope speed, 1250 ft. per min.; motor, 250 hp. with alternating current. In this instance the hoist is 24 ft. horizontally from the shaft center. No thought need be given the proper fleet angle as the hoist can be placed in line with the center line of two shaft compartments.[14] It is easy to calculate the size and holding capacity of the pulley.[19]

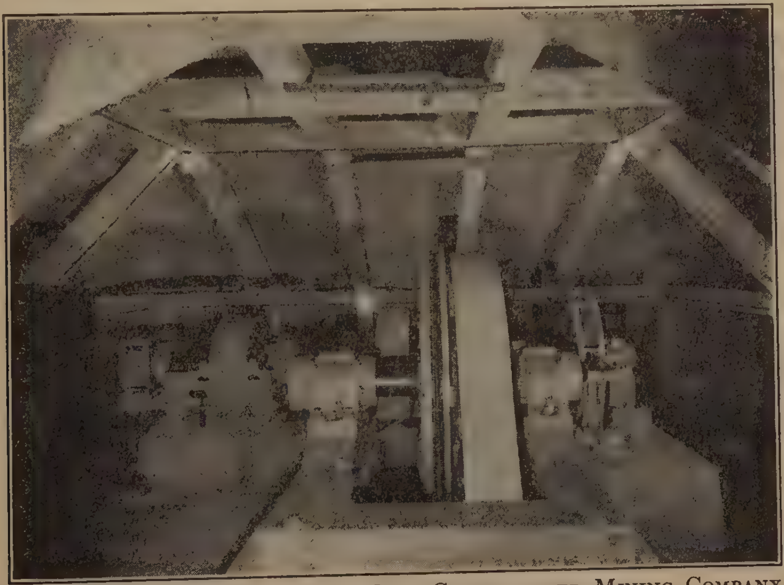


FIG. 7.—BOLLEN PULLEY, PARK UTAH CONSOLIDATED MINING COMPANY. Located directly over shaft and controlled from level below hoist. Motor 250 h.p.; counterweight, 11,000 lb.; skip, 5,000 lb. Hoist built by Nordberg Manufacturing Co.; electrical equipment by General Electric Co.

Other contrivances, such as the Karlik pulley, have been built and many of this type have been installed abroad. In this arrangement, separate clamps are placed on the periphery of the pulley. These are depressed as the rope settles into them and tighten to hold the rope. It would appear that the extensive use of this method indicates that no harmful effect may be expected on the ropes. Its greatest use seems to be in aerial tramways, where great lengths of rope are in use and where rope damage would be expensive. It would appear that this arrangement is less expensive than the Bollen pulley and probably less sensitive to maladjustment.[15]

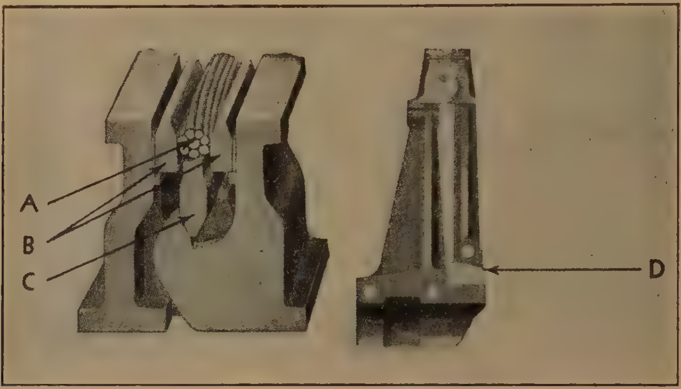


Fig. 8. Fig. 9.
Figs. 8 and 9.—Bollen pulley.
Fig. 8, section through rim; Fig. 9, section through rim and hub.

Another interesting hoist is the Ohnesorge pulley, which is a type of Koepe disk. In this clever arrangement, use is made of several wraps of rope around a drum, thereby increasing the tractive effort. When several wraps are employed, it follows that the rope will tend to travel along the drum as the latter is revolved. By providing a track for the rope, made of links of a chain grooved to fit the rope, and moving this

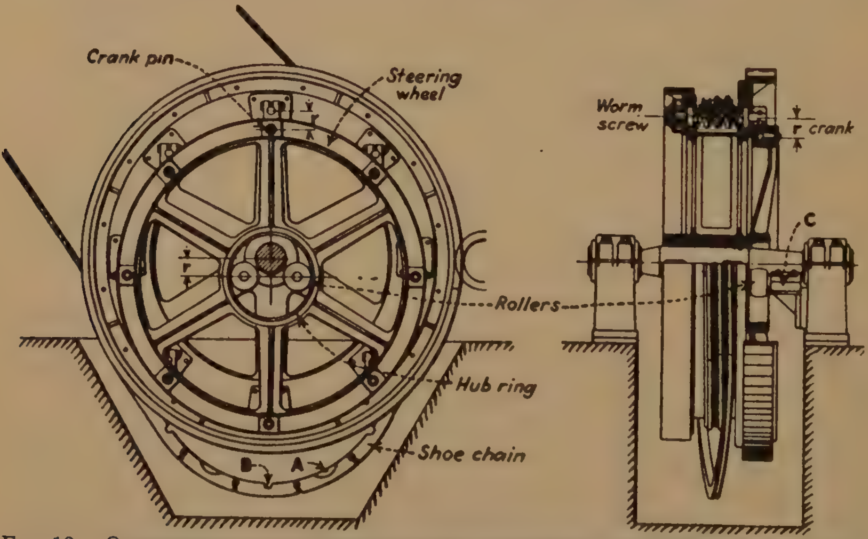


Fig. 10.—Ohnesorge pulley with link chain rope seat and unique worm drive for laterally shifting three wraps of endless chain.

track laterally by means of worms geared to the main shaft, in proportion to the rope pitch, the various wraps remain stationary relatively in the lateral direction. The pulley acts well in operation. This chain, which forms a loop under the drum, travels without vibration or swing. It tends to clatter, which increases with wear of the links. Lateral displacement of the links is said not to cause undue wear of the chain. The hoist under observation was well lubricated, causing an accumulation of grease in the drum pit (Fig. 10). In this arrangement, the ratio of rope pull of the up-going heavy and down-going light rope may be greatly reduced. It is claimed that there is less rope wear, because there is less tendency to slip.[16]

Conclusion

This paper has reviewed some unique and outstanding hoisting equipment for the interest there may be in a special design or a particular hoisting problem. There is a need for special study of each case and no two cases can be identical, except by accident. The mining man has a definite job to do with a hoist and the hoist designer must meet this problem unhampered by existing manufacturing equipment, if possible. He must be free to choose and change designs with little thought to what the problem of the machine shop is going to be, and should utilize the ideas of the mining personnel involved to as great an extent as possible. He should have available shop skill and machinery to carry out his plans precisely and exactly. When the hoisting equipment has been installed and is operating, it usually lasts "indefinitely," as is well borne out in actual practice. The size and shape of the essential unit, the drum, are determined by definite hoisting requirements, power costs and first cost. Hoisting speed has been varied over a wide range and seldom is chosen beyond 4000 ft. per min. Fast acceleration is expensive, but may not always be avoidable. It can be met within limits; for instance, a trial at the Orient shaft, West Frankfort, Ill., indicated an acceleration limit of 14 ft. per sec. per sec. because of the "buckling effect" of the rope of the down-going skip. A hoist should be simple in structure and not complicated by "gadgets," as seems to be proposed for control equipment and braking means. Clutches of either friction or toothed type are usable in large and small hoisting equipment. Good design is essential; good workmanship a definite requirement and experience in both basically necessary.

While good manufacturing is required, the operating and maintenance personnel has had as much to do with the excellent record of hoists as the manufacturer. It is necessary to provide good lubrication and means are today built into a hoist to take care of this requirement effectively. The older hoists have not had this advantage, nevertheless are still operating perfectly. The critical inspection that the maintenance and operating crew gives the equipment and the immediate attention to such details as need it is responsible for this fact. The importance of such care cannot be overestimated.

REFERENCES

1. C. Matschoss: Die Entwicklung der Dampfmaschine, 1927 Edition, 541.
2. B. V. Nordberg: The Compressed Air System of the Anaconda Copper Mining Co., Butte, Montana. *Trans.* A.I.M.E. (1913) **46**, 826.
3. *Engineering* (March 22 and April 12, 1935).
4. C. Legrand: Power Plant of the Burro Mountain Copper Co. *Trans.* A.I.M.E. (1916) **55**, 208.
5. H. K. Burch and M. A. Whiting: Automatic Operation of Mine Hoists as Exemplified by the New Electric Hoists for the Inspiration Consolidated Copper Co. *Trans.* A.I.M.E. (1916) **55**, 10.
6. The Metropolitan-Vickers *Gazette* (Jan. 1937) 322.
7. The Metropolitan-Vickers *Gazette* (Aug. 1937) 36.
8. *Coal Age* (Apr. 9, 1925) 536; (March 12, 1925) 397.
9. H. V. Haight and G. M. Dick: Canadian Mine Hoists. Read at Annual Meeting of A.S.M.E., December 1939.
10. *Dominion Engineer* (April 1937) 3.
11. F. Frölich: Fördermaschine for Boston & Montana Consolidated Copper and Silver Mining Co. V.D.I., 1904.
12. J. Perry and D. M. Smith: Mechanical Braking and Its Influence on Winding Equipment. *Engineering* (Nov. 1932).
13. H. H. Logan: Mine-hoist Control. A.I.M.E. Meeting, February 1940).
14. E. E. Slack: Bollen Pulley Hoist at Magma. *Min. Congress Jnl.* (Feb. 1935).
15. F. Marik: Die Entwicklung der Seilgreiferscheiben mit besonderer Berücksichtigung der Karlikschen Greiferscheiben. *Fördertechnik und Frachtverkehr* (1926) 293.
16. T. Simons and G. Grassmueck: The Koepe Hoist, Equipped with the Ohnesorge Pulley, Invites New Consideration. *Eng. and Min. Jnl.* (Sept. 1934).
17. H. H. Broughton: Electric Winders, 1927 Edition.
18. O. M. Fox: New Orient—The Greatest Coal Mine. *Black Diamond* (Apr. 18, 1925).
19. C. Hanocq: Note sur la poulie Bollen, No. 621851. Univ. de Liége, 1925.

DISCUSSION

(Benjamin F. Tillson presiding)

B. F. TILLSON,* Upper Montclair, N. J.—My discussion is concerned with the mechanics underlying the Koepe and Ohnesorge pulley hoists.

To prevent slippage of hoisting rope on the driving sheave, there must be a definite maximum ratio between the tensions of the loaded and unloaded ropes in balanced hoisting. The ratio S_1/S_2 must be less than $e^{f\pi n}$ where S_1 is tension of loaded rope, S_2 is tension of unloaded rope and the term $f\pi n$ is an exponent of e, which is the base of Napierian logarithms and equals always 2.718; f is the coefficient of friction between the rope and the driving sheave or its lining, n equals the number of *half-laps* of rope about the driving sheave; and π equals 3.1416.

The average value of f for the Ohnesorge pulley is 0.136, therefore the quantity of $e^{f\pi n}$ equals 3.3 where $n = 3$, or 1½ rope turns; and 7.5 where $n = 5$, or 2½ rope turns.

---

* Consulting Engineer.

1. By adopting the correct number of rope turns, high-speed hoisting without rope slippage, and even without a tail rope, may be done in both shallow and deep shafts where the Ohnesorge driving pulley is used.

2. Accordingly, the diameter of an Ohnesorge pulley may be considerably less than a Koepe driving pulley, where only ½ to less than 1 rope turns are available; and its diameter need only be sufficient to keep the bending stresses in the rope below a proper maximum.

3. Because of freedom from slippage, the wear of the rope on the Ohnesorge pulley is far less than that on a Koepe pulley; and the rope is not pinched, as it may be on the Bollen pulley.

4. Perfect lubrication of the rope is possible when it is used on the Ohnesorge pulley because it does not rely upon the high coefficients of friction of Koepe and Bollen pulleys, where about ½ turn is the contact of the rope.

5. Two clutched Ohnesorge pulleys, each carrying a single cage (or skip) with tail rope, make it possible to hoist from various levels in or out of balance. This cannot be done with the plain Koepe pulleys, and the Whiting system of modified Koepe hoists is elaborate. They also permit the discard of fatigued rope ends in the manner usual with drum hoists and the correction of rope stretch.

6. An Ohnesorge pulley is adaptable to the multiple-rope suspension common in elevator systems.

7. The maximum depth of the proposed shaft must be known in order to select a suitable drum hoist; but need not be known for an Ohnesorge pulley hoist, since any greater length of hoisting rope may be used with it.

8. The inertia of driving pulleys is much less than that of hoist drums, so that lower powers of motors will give equal load accelerations with lesser peak demands.

9. Driving pulleys facilitate placing the hoist in the headframe directly above the shaft.

10. The need for a *heavy* tail rope with a Koepe pulley precludes its use in shafts shallower than about 800 ft., but the greater number of half turns on the Ohnesorge pulley overcomes this difficulty.

11. Obviously the ratio of S_1/S_2 is lessened as any increased load is added to S_2 if no greater load is added to S_1. A tail rope adds such a dead load. But the dead weight of empty skip and cage must still be great enough to afford sufficient friction of the rope on the Koepe pulley when the skip is emptying in the dump. Because of its greater number of rope turns the Ohnesorge pulley does not depend on excess dead weight for rope friction and the lighter cages and skips of modern alloys and designs may be employed and thereby reduce the hoisting inertia. For these reasons, even in shallow shafts, no difficulty is experienced from raising a load that rests upon supports in a shaft, such as the "chairs" at a level station or a loading pocket.

The writer feels that greater attention should be given to traction pulley types of hoists whether of the Bollen, Ohnesorge, or other designs. The reduction in initial cost, size of equipment, better proportioning of drive-pulley diameter for rope life, reduction of power demands and cost warrants such consideration when installing new or revising old hoisting plants.

Attention is directed to a novel approach to this problem in the "free-cage system" illustrated in Fig. VII-31 on page 57 of the Mine Plant volume of the A.I.M.E.

H. V. Haight* and G. M. Dick,† Sherbrooke, Quebec.—While we recognize the peculiar advantages of both positive and friction clutches, we cannot agree with Mr. Nordberg's statement that positive clutches do not permit setting the drums to the desired level difference necessary, thus necessitating maneuvering of the hoist to bring cage or skip to the level of the station. Our experience has been that the rope-stretch difference that exists in a rope when the skip is loaded, as against when the skip is not loaded, will far exceed any slight variation due to pitch of clutch teeth. Then again, when cages are used these almost invariably sit on "chairs" when they reach the stations, so here again a slight difference in length of rope is unimportant. Further, it must be kept in mind that the difference in diameter at which the hoisting and lowering ropes are frequently winding makes absolute station registration impossible. Therefore we rather doubt the accuracy of Mr. Nordberg's statement regarding the reason why friction clutches were adopted.

---

* Consulting Engineer, Canadian Ingersoll-Rand Co.

† Chief Hoist Engineer, Canadian Ingersoll-Rand Co.

Our experience rather suggests that the main reason why friction clutches were introduced was their ease of engagement as compared with the older designs of positive, or toothed clutches, where almost absolutely correct spotting was necessary. This advantage in ease of engagement is not now restricted to friction clutches, however, since the introduction of the internal expanding positive clutch with V-shaped teeth.

Another peculiarity inherent in friction clutches is that they can be engaged when the hoist is running, which some operators feel is an advantage, claiming that in the case of brake failure it is possible to engage the clutch and use the brake on the opposite drum to bring the cage or skip to rest. This possibility, however, is certainly subject to many dangerous consequences, unless accomplished by an expert and skillful operator.

The slippage of which friction clutches are capable is sometimes considered an advantage when an obstruction accidentally causes sudden stopping of cage or skip in the shaft. The slippage in this case relieves the shock which otherwise would come on the hoist and rope. This same characteristic of "slippage" is considered by another school of thought to be a dangerous disadvantage, especially when operating at high speeds, because of the possibility it gives to the cages or skips to get out of correct relationship with one another; the latter view of the matter being responsible, we presume, for the fact that friction clutches have not been popular in South Africa on large high-speed hoists.

We feel that Mr. Nordberg would have been more nearly correct had he stated that the selection of the type of clutch was dependent on the peculiar conditions of operation, which were anticipated, and that on very large hoists the excessive size of the component parts of a friction clutch, together with its proportionally large WR^2 made serious consideration of the alternative use of positive clutches an obvious necessity, especially when positive clutches of modern design embraced so many advantages, previously provided only in friction clutches.

We heartily agree with Mr. Nordberg in his comments on brake controls and his advice that complications in this connection should be avoided. We regret that he did not take the space to give his views on the relative merits of proportional pressure braking as compared with the proportional travel system. We feel sure he could have given some interesting comments on the newer of these two alternatives.

Mr. Nordberg's concluding paragraph touches one of the most important phases of mine-hoist building, in that he infers that a large part of the success of a mine-hoist installation depends on the experience the hoist designer has acquired from previous hoisting problems and from which he can readily provide important recommendations for new equipment. In other words, it requires more than good machine-tool equipment to produce successful hoists; it rather depends largely on the engineering knowledge and experience of the designer as acquired only by years of close study of this very interesting subject.

B. V. E. Nordberg (author's reply).—There is no doubt of the advantage pointed out by Mr. B. F. Tillson in the discussion of the Bollen and Ohnesorge arrangements.

Two-stage hoisting—that is, lifting loads in steps—that requires placing the hoist underground requires very inexpensive excavation to house the hoist, an advantage already pointed out in the paper.

The disadvantage of the Ohnesorge type, as compared with the Bollen type, is in its first cost, noisy operation and the requirement of copious lubrication under the chain and on the worm gear; it is more complicated and its upkeep is still an unknown quantity. These disadvantages are mentioned to balance the advantages enumerated by Mr. Tillson, and should be carefully weighed. Both Bollen and Ohnesorge types should offer much that mining men have been looking for.

The clutches of the tooth type referred to by Messrs. Haight and Dick are, of course, those used on small hoists where a tooth-pitch diameter is not much greater than the drum diameter, and this was considered in the paper. It is such hoisting drums of large diameter that are much more critical as to "spotting" the cage or skip correctly that are considered. Such applications as shown on Fig. 6 illustrate a large difference in the relative diameter of the drum and the clutch, so that even the use of fine teeth still causes a relatively large movement on the rope. It also points the way to using a small-diameter clutch for a large drum and is an illustration of economical use of metal. It should further illustrate that if such economy were used on smaller hoists, a proportionate difference per tooth displacement would mean a considerable movement on the hoist rope. Perhaps the friction type of clutch, with its advantages, made necessary the use of tooth clutches of larger diameter. Mr. Haight's diplomatic expression of the reasons for the development of the tooth clutch is broad enough to cover the writer's more crudely expressed reason which was based upon a statement of a mine operator in South Africa. Undoubtedly there is some historical expression of this fact, but at the moment it is not at hand. Be that as it may, both Canadian and American manufacturers are governed by requirement and specification.

The advantage of the friction clutch used as a brake in case of mismaneuver is of doubtful value. Such use has been made and is of record, but correct interlocking between clutch and brake application should avoid any accident that would necessitate such use. The writer has however witnessed a shaft inspection where the inspectors were lowered on a cage, unclutched from the engine (steam being used in that instance), and the brake used to lower on. Any accident to the brake would have meant the necessity for use of the clutch and no ordinary interlocking, in that event, would be possible. Such operation should be discouraged.

The discussion by Messrs. Haight and Dick has brought out some new points and has strengthened some of the statements of the paper.

An Experimental Study of the Rock Drill

By Tomizi Suzuki,* Member A.I.M.E.

(New York Meeting, February 1939)

Digest by B. F. Tillson†

Labor conditions have finally caused rock-drilling machines to supplant hand drillers in the Japanese mining industry, and have encouraged this study. During a period of three years 25,000 tests were made on 67 different American, Japanese, and European rock drills, using, respectively, the Paynter testing machine, and drilling in granite and limestone. Recorded were: air consumption, speed of rotation, torque of rotation, foot-pounds of striking energy, air pressures, input horsepowers, and speed of rock penetration. The author expresses his views on the interrelation of these factors and offers theoretical support, as well, by formulas based upon the mechanics of impact. He discusses the different designs of the valves, cylinders, piston hammers and rotation devices involved in the different types of rock drills, with respect to their merits and defects and offers conclusions about the factors that should be considered in the new design of a superior rock drill.

The author's mathematical demonstrations and arguments for a value of "best feed clearance" of the drill bit from the rock are based upon the hand-operated screw-fed type of hammer drill. Automatic feeds have not been considered in this paper except for the tests on one drifter with air feed, 10 air-feed stopers, and some hand-held blockholers. The following classes of drills, with one of each model, size, or type, were tested: 23 heavy drifters, 34 light drifters and hand-held drills, and 10 stopers. They represented the following makes and models: 40 from the United States—namely, Ingersoll-Rand 18, 148, L-74, N-70, N-72, N-75, R-72, 26, BAR-33, BCR-430, DCRW-23, RA-12, CA-31, CC-11; Denver 7, 107, WT-21, WT-31, WT-331, WT-337 (models WT with motor rotation, screw feed), 11, Waugh 93, 95; Waugh 16; Sullivan DR-6, DW-64, T-3; DP-321, DP-331; DA-21, DT-44, DT-421; Gilman AX-321 (motor rotation and air feed); Cleveland D-5; AIBQ, 44-SWR,

---

* General Manager, Hitachi Mines, Nippon Mining Co., Tokyo, Japan.

† The entire paper (A.I.M.E. Contribution 111) has been filed with the American Documentation Institute, 2101 Constitution Avenue, Washington, D. C. It may be obtained from that Institute by ordering Document 1179 and remitting 73¢ for copy in microfilm (read enlarged to full size on reading machines now widely available), or $5.50 for copy in paper photoprints legible without mechanical aid.

44-TWR, 44-TDR; S-3D; Chicago P. T. CO. CP-5; 5 drills from England—Holman PV-160; PL-50 Pdr; 17 Pdr; AFW-2-W; 5 drills from Germany—Demag HBK-70; NHK-60W, 65-W; NHK-65 PV; 5 drills from Sweden—Atlas RW-64; BOB-8, 10-D, 10-W, 12; 4 drills from Austria—Bohler BH-65, 140, 200, H-62; 2 drills from France—Meudon AP, BP; and 6 drills from Japan—Ashio 11, 12, 30; Besshi; Sonoike, Yamayu; Hitachi, 7.

The author found many opportunities for error in the use of the Paynter tester to determine the frequency and impact energy of blows delivered from a rock drill to a drill bit. He took steps to overcome them. Of importance was the use of an oil, having the proper temperature-viscosity characteristics, suitable for the pressure transmission of impact blows. By 800 tests on 22 kinds of oils he found 8 suitable for use, including a seed oil and a petroleum oil.

From tests and mathematical derivations he decides that an important factor in the effective transfer of energy from a rock-drilling machine to the rock is the normal distance of the drill bit from the medium that it strikes. This distance he calls feed clearance. His tests support the best distance of about 3 mm. mentioned by B. F. Tillson,* but will vary with the material drilled, the size and type of rock drill used, and possibly the weight of the drill steel. This feed clearance also influences the action in the rock drill: the energy per blow, the number of blows per minute, the rotation, and the air consumption.

He quotes formulas of elastic impact of bodies, and with symbols:

V, velocity of piston hammer just before impact.

W, weight of piston hammer just before impact.

w, weight of drill steel.

e, coefficient of restitution (for steel e equals 0.66).

g, acceleration due to gravity.

k_1 = coefficient of loss of kinetic energy with impact of hammer on drill steel $= \dfrac{(1 - e^2)WwV^2}{2g(W + w)}$

k_2 = coefficient of effective hammer energy transmitted to rock $= \dfrac{(1 + e)^2Ww}{(W + w)^2}$

Depending upon the size of drill steel used with a given rock drill, k_1 varies from 25 to 42 per cent, and k_2 varies from 50 to 70 per cent.

The original paper is replete with extensive tabulations of test data, graphs, diagrams, and formulas, which cannot be reproduced in this digest but which will be made available as microfilm (see footnote, page 206). Facts, opinions and conclusions of the author are therefore recited here in concise statements, numbered for reference. This paper does not give consideration to the transmission of energy vibrations as compressural waves through the drill steel to the rock.

---

* *Trans.* A.I.M.E. (1921) **66**.

AUTHOR'S STATEMENTS

1. Spool valves gave good results on both heavy and light drifters or hand-held blockholers, especially the latter two: tubular and flapper valves were next best, the former good in heavy drifters and the latter in both heavy and light drifters; butterfly and arc valves have proved entirely unsuitable for heavy drifters; valveless types of stopers are superior and the shell valve type are inferior, valveless types can be used in heavy drifters but the results are not then very good.

2. The hardness of the cylinder should be Rockwell C-60.

3. In Japan the preferred weight of rock drills is 121 lb. (55 kg.).

4. There is a "best feed clearance" of the cutting bit from the rock, to facilitate rotation and ejection of cuttings and develop maximum output horsepower of the rock drill.

5. The most effective length of drill steel has a weight equal to the weight of the piston hammer. The author gives mathematical derivations in support of this. They are derived for a rifle-bar rotation rock drill but he says they will apply as well to the independent motor-rotation rock drill.

6. The measured depth of chipping grooves in rock and correspondingly desirable "feed-clearance" are: 4 mm. in granite and 3.5 mm. in limestone when using a heavy drifter; 2 mm. in granite and 4 mm. in limestone when using a light drifter. From sizing tests of the rock cuttings the author concludes that the "best feed clearances" should be at least 4 to 8 mm. with the heavy drifter and 4 to 6 mm. with the light drifter.

7. The drilling speed of the Carr bit is 80 to 95 per cent of that of the cross bit, and its inferiority shows particularly in soft rock. The best angle between the planes of the cutting edges of a bit is 90° for hard rock and more acute for soft rock.

8. The heavy drifter is suitable only for rocks harder than granite wherein the energy per blow is too great and the number of blows per minute too few; the stoper type is best for limestone, but for softer rocks the light drifters and hand-held blockholers are advantageous.

9. With a 2-in. (50-mm.) gauge drill bit the necessary energy per blow is 18.082 ft-lb. for soft rocks and 36.165 ft-lb. for hard rocks. The energy of blow should vary directly as the diameter of bit gauge.

10. The drilling speed of rock penetration varies inversely as the square of the diameter of bit gauge.

11. A small angle of rotation per blow is better in hard rock and a large angle in soft rock, up to 90° angle for cement. The best rotation angle also varies with the power of the blow.

12. The independent rotation type of rock drill showed no special difference in speed of rotation when compared with the dependent rota-

tion (rifle bar) type. The latter averaged 200 r.p.m. and 1600 blows per minute, or 8 blows for one revolution of rotation.

13. The rotation horsepower of the heavy drifter is about four times that of the light drifter and blockholer, and the independent rotation has 1.55 times the horsepower of the dependent rotation heavy drifter. This ratio shows larger at low air pressures of 57 lb. per sq. in. (4 kg. per sq. cm.). The maximum value for independent rotation was 0.479 hp., the minimum value, dependent rotation, was 0.035 hp. with heavy drifters. The maximum value for light drifters, dependent rotation, was 0.086 hp.

The torque of rotation for the heavy drifter is about five times that of the light drifter; and the independent rotation 1.81 times that of dependent rotation.

This proves that the independent rotation type drill is better suited for deep drilling.

14. The horsepower for air compression must be estimated at 3 to 5 times the air indicated horsepower of the rock drill. Based upon the output horsepower of the rock drill, as shown by the Paynter tester, and the compressor horsepower required, the over-all efficiency can be expected to fall in the range of 4 to 8 per cent, and was only 3 to 5 per cent with the compressor used by the author.

The distance drilled per compressor horsepower was: 0.374 in. (0.95 cm.) for heavy drifters, 0.24 in. (0.61 cm.) for light drifters and hand-held blockholers, and 0.393 in. (1.0 cm.) for stopers.

15. Owing to differences in sizes and design, the maximum output horsepower had a ratio of 3 to 1 compared to the minimum in tests of heavy drifters. With the light drifters and blockholers this ratio was nearly 7 to 1; and with stopers the ratio was 7 to 1.

The over-all efficiency of the stoper is usually comparatively high. Its maximum by Paynter tester for air indicated horsepower was at a pressure of 85.34 lb. per sq. in. (6 kg. per sq. cm.), 37.6 per cent, or at 99.56 lb. per sq. in. (7 kg. per sq. cm.), 38.2 per cent; and the corresponding over-all efficiencies based on compressor horsepower were 12.6 and 12.1 per cent. The minimum over-all efficiencies of the tests were 3.2 per cent, found with several light drifters and blockholers.

Tillson's published Factor of Desirability equal to the quotient of drilling speed in inches per minute divided by the free air consumption in cubic feet per inch drilled is endorsed by the author's tests. The same formula may also be expressed as the quotient of the square of the inches drilled divided by the product of the number of drilling minutes times the cubic feet of free air; or may be expressed as the quotient of the square of the drilling speed per minute divided by the air consumption per minute.

16. The efficiency of rock drills usually increases with an increase of the pressure of compressed air until the latter reaches 99.56 lb. per sq. in. (7 kg. per sq. cm.). The author's experimental data agree in general with formulas E. M. Weston.

17. Usually a flute angle of 5° is adequate for the rifle bar of a dependent rotation rock drill. For better spool-valve efficiency it is necessary to enlarge the valve diameter and shorten its stroke, and the rear of the cylinder is a favored place for the location of spool, circular, and tabular valves. The butterfly valve has been supplanted by the rectangular flapper valve, and it in turn by the circular valve. In the ideal rock drill the energy per blow is limited to that needed to chip the rock and the number of such blows is increased. Within such limitations the lighter the piston hammer the better. Increase in stroke involves an awkward increase in the length of the drill, so an increase of cylinder diameter is indicated. For a given weight of piston hammer its increase in diameter reduces its length of bearing in the cylinder and then a longer washer bearing must be provided to prevent air leakage and ensure an air cushion.

Author's Conclusions for Design of a New Rock Drill

18. Regulate the energy of the blow to the texture of the rock and increase the number of blows per minute to the point where the fixed energy of the blow would be detrimentally affected.

19. Enlarge the valve diameter, decrease its total weight and stroke, and increase the number of blows per minute.

20. Increase the diameter of the piston hammer, decrease the cylinder bearing to the minimum but make the washer bearing considerably longer; decrease the weight of the piston hammer as much as possible so as to increase its speed, thereby increasing the energy per blow and the number of blows per minute.

21. Decrease the flute angle of the rifle bar as much as possible without cutting down the rotation speed, in order to increase the number of blows per minute by lessening the friction in the rotation mechanism.

22. The lighter the rock drill is in weight, the better it is. An elimination of vibration and a decrease in the number of its parts are essential for its practical application. Decrease both the length and the height of the rock drill and promote its handling by the absence of all protruding parts. Superior materials should be used in its construction and light special alloys are likely to be introduced in the construction of the valve, piston hammer, and other parts.

Hot-milling of Rock-drill Bits

By Robert C. Berggren*

(New York Meeting, February 1940)

The hot-milling process for reconditioning rock-drill bits is not new. It has been employed by a few mines for years and in the past decade it has been widely adopted, as its advantages have become better known. Because of the extensive commercial development in England and Canada of machines to operate on this principle, general acceptance has been largely in the British possessions and until recently has been chiefly in the field of conventional steel. In the past two years there has been a definite trend toward hot-milling of detachable bits in Canada,† but only in the past year has any considerable attempt been made to build special machines for this type of work. Generally a machine designed for conventional steel has been used and special holders employed to manipulate the bit.

As early as 1934 at least one operator in the United States had adapted hot-milling to conventional steel with a substantial saving in resharpening costs.[1] Several other operators have taken it up but there has been no widespread adoption of the method in this country by users of conventional steel. This may be partly explained by the fact that until recently no American manufacturer produced this equipment and operators have been obliged to build their own hot-millers or import them. Several years ago a few machines were built in Seattle, Washington, and operated in some mines in the northwest, but, for reasons not generally known, the effort was abandoned.

The Process

Hot-milling performs the function of re-dressing the bit at a forging temperature of about 1650°F., by the removal of metal from the flutes with a milling cutter, followed by a similar cut from the bit gauge. Conventional bits are thus treated following the usual forging operation in a sharpener, and usually in the same heat.

Manuscript received at the office of the Institute Jan. 22, 1930. Issued as T.P. 1215 in Mining Technology, September 1940.

* Engineer for The Mine and Smelter Supply Co., Denver, Colorado.

† Included in Canadian mines that have adopted hot-milling to removable bits are: Aldermac Copper (Quebec Corporation, Limited); Asbestos Corporation of America, Limited; Coniaurum Mines, Limited; Lapa Cadillac Gold Mines, Siscoe Gold Mines, Perron Gold Mines.

[1] References are at the end of the paper.

Removable bits are hot-milled in essentially the same way, except that the die-forging operation is not employed to swage the bit to approximate gauge. Normally a standard removable bit is used of heavy body design and the removal of metal by the cutter is somewhat analogous to the grinding method in that the cutter and grinding wheel perform the same duty, and the same surfaces of the bit are brought in contact with the cutting device.

A hot-milling machine consists of a motor-driven rotary milling cutter suitably mounted on a main frame or base. Guides and stops confine the bit to the desired position in relation to the cutter. The bit is milled in two main operations, which successively bring the bit wings to sharpness by the "fluting cuts" and take off the gauge to the desired diameter by the "gauging cut."

Thus far the operating principles as described may apply to both conventional and removable bits. Specific consideration, however, of the possibilities of hot-milling at once make evident certain inherent differences in the reconditioning procedure of removable and conventional bits in the direction of the ideal solution of the principles involved.

The process as applied to conventional steel constitutes a clean-up operation after reforging in a sharpener. The chief advantages as related to drilling and reconditioning economy are:

1. Decreased sharpening costs[1] due to: (*a*) saving of labor through the over-all shortening of the sharpening cycle; (*b*) longer effective use of sharpening dies; (*c*) low maintenance cost of hot-miller. Bits ordinarily considered defective resulting from worn dies may be corrected in the subsequent application of hot-milling. The sharpener more or less "roughs out" the bit.

2. Removal of scale after forging, thus enabling correct tempering and hence longer bit life in the hole.

3. Gauge uniformity of machined quality and consequently greater drilling efficiency.[2,3]

Removable bits are completely resharpened in the hot-miller and receive this treatment immediately after being removed from the furnace. Economic consideration of hot-milling hinges on two main points, and is stated in comparative terms of the equipment it either stands to augment or replace: (1) more re-uses per bit; (2) greater bit production per unit of time.

Uniformity of the finished bit cannot properly be included in the two points above, but inasmuch as some of the better grinders maintain very close tolerances in the process of regrinding, it appears that the complete control of these important dimensions with comparative ease on a hot-miller is far overshadowed by these more important aspects, and the fact is simply mentioned at this point.

As a distinctly new field with apparently considerable possibilities, the recent surge of interest and activity in the hot-milling of removable bits has been accelerated by a growing recognition of these possibilities and has resulted in an effort to relegate it to "system" adaptiveness, and to find a mechanical solution. New developments in this field have been so rapid as to defy complete intelligence of the widely scattered activity in the short period of time for which deducive study has been possible. It appears to the author, however, that a comprehensive approach to developments that are generally known, and a detailed account of some of his observations gathered through direct contact or by competent reporters, should sum up a few of the more important results.

DEVELOPMENT TRENDS

The several different machines for removable bits that have evolved in the past year differ in principle chiefly in the manner in which the bit is presented to the cutter or cutters. At present there appear to be two schools of opinion as to the most effective and efficient treatment of the bit in the "fluting cut" previously described. One school favors the "receding cut"* whereas the other school claims advantages for the "approaching cut"* in the fluting operation that brings the wings to sharpness. The two schemes may produce bits similar in appearance, yet several vital factors in speed of operation and the number of re-uses obtainable are affected.

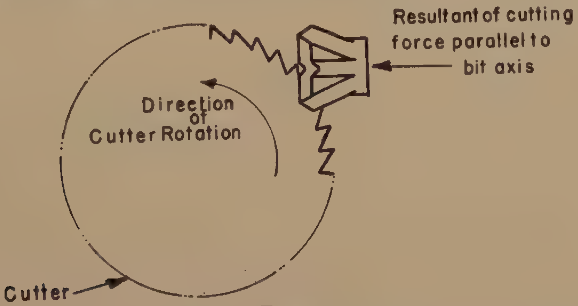

Resultant of cutting force parallel to bit axis
Direction of Cutter Rotation
Cutter

FIG. 1.—RECEDING CUT.

The "receding cut" (Fig. 1) is subject to a component of force that tends to remove the bit from the stud, which is inserted in the threaded end and which holds the bit against the cutter. This condition necessitates a holder, which must be expanded in the threaded hole or in some similar manner firmly grasp the bit.

In the "approaching cut" (Fig. 2) the forces that are generated during contact in cutting tend to push the bit back on its holder. This condition obviates the use of a special holder, and the bit may simply be placed on a plain collet integral with the machine and after milling removed without the extra operations of locking and unlocking a holder. The advantages of the approaching cut for the sake of production are thus self-evident and fortunately the same forces that simplify the bit manipulation are responsible for further economies by the introduction of the so-called "upsetting action," which delivers a net gain of gauge on the finished bit. Such gauge advantages are extremely important in their influence on the added effective life of the bit.

---

* The author takes the liberty of paraphrasing in somewhat more simple language a rather involved technical distinction which is treated in the text.

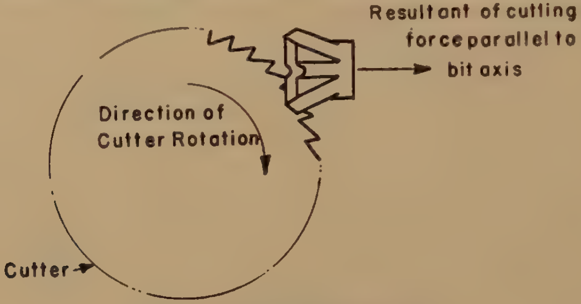

Resultant of cutting force parallel to bit axis
Direction of Cutter Rotation
Cutter

FIG. 2.—APPROACHING CUT.

The gauging operation (Fig. 3) is performed by rotating the bit (still hot) against a rotary milling cutter. Further divergence of principle is encountered in the mechanical application of this, the second and last main operation. The "single cutter" method (Fig. 3*A*) employs the same cutter for fluting and gauging, whereas the "double cutter" scheme (Fig. 3*B*) uses separate cutters for each of these steps. Machines for conventional steel have long employed the single cutter, and little additional advantage is seen through the addition of a separate gauging cutter for this particular undertaking. The different handling characteristics of the removable bits evince other possibilities for the gauge cut and upon these factors are based the deviations in design. The manually operated holder, mentioned in the paragraph on the receding cut, is manipulated in much the same way as a conventional drill piece, therefore disciples of this "school" are the principal advocates of the single cutter.

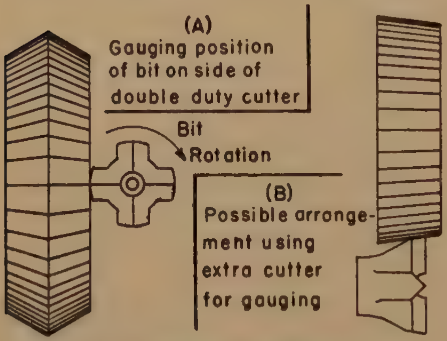

(A)
Gauging position of bit on side of double duty cutter
Bit Rotation
(B)
Possible arrangement using extra cutter for gauging

FIG. 3.—GAUGING ARRANGEMENTS.

The "double cutter" scheme may or may not be used where either type of fluting cut is used. However, with the elimination of the holder that is possible in the approaching cut, the bit may be taken by the machine from the operator's tongs or pliers and completely milled quite simply. The extra cutter simplifies the mechanical motion and is readily adjusted to angle and gauge size. There appear to be advantages with both of these gauging possibilities and in general it may be said that compactness and first cost favor the double-duty single cutter while production speed and adjustability favor the use of two cutters.*

OPERATION AT A WESTERN MINE

Some interesting data have been obtained from the hot-milling practice of a large western mine. The following account is based on material gathered during a period when the work was of an experimental nature and periodical changes that required intermittent operation interfered with the acquisition of a set of complete figures and costs.

The hot-miller used was a double cutter with approaching cut. One operator and a furnaceman were used for hot-milling. Furnace facilities were not adequate for uninterrupted treatment and tempering was done during the following shift. The following method was used:

1. Dull bits were assorted to ⅟16-in. gauge sizes and transferred from point of deposition to sharpening shop in metal boxes 14 by 14 by 14 inches.

2. Starting with larger sizes dull bits were loaded face down on a furnace-loading tray 14 by 15 in. The bit load was 56 to 72, depending on size.

3. Bits were loaded into one side of the furnace and 15 min. later the other side was loaded. The bits were permitted to come up to 1600°F., which required from 30 to 35 min. The furnace used was an oven type, electric, 61 kw., 220 volts.

4. When up to heat the bits were extracted from the furnace with a right-angle poker and placed on a brick-topped table, 48 by 72 in., next to the hot-miller.

5. A bit was picked up from the table by the operator with a light pair of 10-in. offset pliers and placed on the collet for milling. The hot-milling operation consisted of four rough fluting cuts in which full engagement against the stop sharpened the wings and three light finish cuts relieved the burrs on the cutting edges. The metal was instantly removed and not over one second was required for each of the four main cuts.

---

* Confusion may arise from the fact that a good many conventional-steel hot-millers employ the use of two cutters, and it has been pointed out that these machines are equipped with cutters designed for both flute and gauge cuts separately. When two cutters are used on a conventional-steel hot-miller it is for the purpose of increasing production and two operators are used simultaneously and the operation is complete on either cutter.

After fluting, the fixture was swung over to engage the gauging dolly, which rotated the bit against the gauging cutter. Removal of the bit from the collet was sometimes followed by a reaming operation in which the reopening of the water hole with the reaming attachment finished the hot-milling operation. The reaming attachment was on the side of the machine and was used on short sizes in which the water hole became more confined near the bottom of its taper, and burring required reopening of the hole. The bit was then tossed back to the table, on which were metal separators for each gauge of bit, and permitted to cool.

6. For hardening, the bits were loaded on the tray as explained in step No. 2, and heated to 1430°F., which required 30 to 35 min., then they were extracted from the furnace and placed on a rotary quenching unit. This unit consisted of 12 cups of 3¼-in. diameter, rotated by a ½-hp. motor through a 180 to 1 reduction drive. A complete revolution required 90 sec. The quenching medium was water held at 100° to 110°F. by a thermostatic control and a magnetic cold-water valve. The cup screens were set to 3⁄16-in. quenching depth and were adjustable. After rotation through the quenching cycle, the bit was automatically tumbled off.

7. Since all bits were separated for gauge size after milling this order was maintained during the tempering and the bits were later loaded into the proper metal containers correctly marked, and were thus ready for distribution.

This mine requires 1750 resharpened removable bits per day. When the sharpening was done by regrinding, considerable trouble arose from the cracking of bits, although the bits were annealed before grinding, in order to prevent cracking.

Upon installation of the hot-miller, about 900 bits per day were reconditioned by this method when operation was uninterrupted. There was the problem of converting the arrangement that had been used for the regrind method over to hot-milling, with a consequent delay in production, part of which was occasioned by the need of an extra furnace. It was necessary to shut down the hot-miller during the hardening operation, since one furnace could not do both jobs at the same time.

It was estimated that the cost of $0.035 per bit reconditioned on a grinder would be reduced to $0.020 by a production of 400 to 450 bits per man-shift on the hot-miller. This was based on a power rate of $0.0075 per kilowatt-hour and a wage of $5.25 per man-shift. The accuracy of this estimate cannot be established until the contemplated changes previously mentioned have been made.

The advantages of hot-milling over grinding at this mine were found to be:

1. Increased production per man-shift. Two men ground and one man hardened 700 annealed bits per shift. Production was 233 bits per

man-shift. Hot-milling bit production with the installation of the tempering furnace will be about 450 bits per man-shift.

2. Greater number of uses. About 30 per cent more uses were obtained with hot-milled bits. By test more than 50 per cent of the starters were found to come back to their original gauge.

3. Uniform flute and gauge dimensions. Milling cutter wear negligible with respect to shape of cut. Adjustment of gauging cutter easily made to 1/64 inch.

4. Reclaiming of mushroomed* and chipped bits. A considerable number of defective bits that had been discarded were put back into service. Reclaiming on a grinder consumed time and meant considerable wheel wear. Mushroomed bits were reconditioned with no extra effort. Chipped bits were reclaimed when such defects would clean up with the cutter.†

5. Hazard of burning and cracking bits was eliminated. (Subsequent faulty hardening may cause cracking in any event and the comparison is made in connection with defects of this nature originating in the grinding operation.)

Eastern Customs Shop

An eastern shop has successfully employed hot-milling in the reconditioning of removable bits. The production figure is more impressive than that shown in the foregoing example of the western mine.

The hot-miller was a double cutter with approaching cut, similar to the one used at the western mine discussed. Conditions were more favorable for efficient operation, however, and the auxiliary equipment of furnaces and heat-treating apparatus were well suited for the work. The quality of material turned out by this shop has always been exceptionally high, and a constant effort has been made to produce a better reconditioned bit under profitable circumstances. The operators are higher paid and somewhat more intelligent than the average operators assigned to this job in the field. On certain types of bits production has been increased to 1800 reconditioned bits in an 8-hr. shift with the hot-miller. An operator and furnaceman comprise a crew. In this instance, the bits were for a shank of 1¼-in. diameter and, although of a large size, the reaming operation was not necessary, which contributed somewhat to a saving in time.

---

* Mushrooming is not uncommon. Excessive pressure against the wheel may produce temperatures in the bit which are destructive to hardness with subsequent flattening of the cutting edges upon impact in the hole.

† It is reported that another western mine has reclaimed by hot-milling 20,000 bits that had accumulated because of regrinding defects, and which could not be profitably retreated on a grinder.

A further favorable aspect of hot-milling was found in the additional number of re-uses. On one eastern construction job only about half of the starters were cleaning up to seconds on a grinder, the remainder suffering excessive gauge loss and being finished at the third gauge size. The sequence of gauge sizes were 2¼ in., 2⅛ in., 2 in., 1⅞ in. and 1¾ in. The starter was taken in 36 in. and the successive changes were made at intervals of 24 in. The application of hot-milling brought all of the dull starters to second gauge and made unnecessary the stocking of new bits for half the total quantity used in that size.

ADVANTAGES OF HOT-MILLING

The advantages of hot-milling of removable bits may be summed up in the following points:

1. Production of refinished bits is substantially increased on the "per man" basis with the proper application of hot-milling.

2. Maximum machine production does not retard the quality of the finished bit, a feature that places a definite limiting point on all grinders. Cracking and burning are not limiting factors to production.

Successful application of the re-grind method, as contemplated in the preceding paragraph, is limited to the quality of the bit obtained. The production possibilities of hot-milling are limited only to mechanical considerations and the effort of the operator.

3. More re-uses are obtained. This most important single point should be good news to bit manufacturers and users alike. A few figures have been submitted by the author to substantiate this claim and more extensive research would be necessary and under conditions more compatible to the time factor in order to present this matter in more detail.

GRINDING VERSUS HOT-MILLING

Hot-milling at the present time and in its existing state of development is best suited to the large operator. Increased bit production with a hot-miller requires a machine that will measure up to the possibilities and the cost of this would be prohibitive to a small operator. Auxiliary equipment such as a furnace and quenching unit necessary for this treatment further confine hot-milling to a definite production bracket.

From all present indications the grinder will continue to serve best the needs of the smaller operator who reconditions his own bits. The advantages of hot-milling are completely active only in the region of production that would be difficult or impossible to attain with a grinder on a "per man" basis. The benefit of greater production of reconditioned bits and more re-uses per bit must be deducted from the larger investment in equipment and the maintenance of the auxiliary units.

It is predicted that hot-milling will bring into existence a number of new custom bit-sharpening shops in communities where they may be of service to several small operators. Such shops will owe their origin and existence to the assumption that a superior reconditioned bit can be supplied for substantially the same and possibly less cost to the operator than he can obtain with other equipment. This activity probably will be confined to removable bits, since transportation problems and possibly an increased investment for circulating stock would tend to discourage such an arrangement with conventional steel.

REFERENCES

1. McH. Mosier: Progress Rept. No. 4 on Investigation of Detachable Rock-drill Bits. U. S. Bur. Mines *I.C.* 6951 (1939). 37 pp.
2. McH. Mosier and W. G. Agnew: Bit Gage vs. Drilling Speed. U. S. Bur. Mines *R. I.* 3447 (1939). 4 pp.
3. L. Eaton: Refinements in Design of Rock-drill Bits. A.I.M.E. *Tech. Pub.* 1095 (*Min. Tech.* Sept. 1939).

Recent Trend in Drill-steel Gauge at Homestake

By Harlan A. Walker,* Member A.I.M.E.

(New York Meeting, February 1940)

Rock-drill steel has an important bearing on costs in many mining operations, both directly and indirectly. Direct factors include such items as shop expense, steel consumed per ton of ore produced, cost of distribution and the dust hazard. The latter, although no specific amount may be assigned to it, is an ever-present problem, and is receiving increased attention as the science of mining progresses. Indirectly drill-steel performance affects consumption of compressed air, maintenance of drills and drilling speed. The latter, in turn, increases or retards the tempo of mining and development. These factors, while not all inclusive, are preponderant in their importance. Thus it is evident that in an extensive operation drill-steel expense may run to many thousands of dollars and that thoughtful research leading toward more efficient practice is warranted.

Experimental Work

During the late spring of 1939, experiments were undertaken at Homestake in reducing changes in drill-steel gauge from 1/8 in. to 1/16 in.

Table 1.—*Comparative Bit Changes*

Drill Lengths, Ft.	Homestake Standards, In.	
	Before 8-1-39	After 8-1-39
Starters	2 3/8	1 7/8
3	2 1/4	1 13/16
4	2 1/8	1 3/4
5	2	1 11/16
6	1 7/8	1 5/8
7	1 3/4	1 9/16
8	1 5/8	1 1/2
9	1 1/2	1 7/16
10	1 3/8	1 3/8

In drifts and crosscut headings mounted machines were set up and holes were drilled with 1/8-in. bit-gauge changes and other holes a few inches

---

Manuscript received at the office of the Institute Jan. 24, 1940. Issued as T.P. 1214 in Mining Technology, July 1940.

* Assistant General Manager, Homestake Mining Co., Lead, South Dakota.

away with $\frac{1}{16}$-in. bit-gauge changes, thus minimizing local variations in rock texture and hardness. In raises, a sufficient number of alternate rounds were run with the old type and new type of bits to assure an average condition. At first it was believed that the hardness of the ground would necessitate a second series of experiments with $\frac{3}{32}$-in. reductions, but from the outset the $\frac{1}{16}$-in. standard was so successful that experiments were made in the drilling of standard rounds with the new steel. As a result, in June work was begun on changing over all steel to the new gauge and on Aug. 1 it was adopted throughout the operation. Comparative changes are listed in Table 1.

Type of Ore and Rock Formation

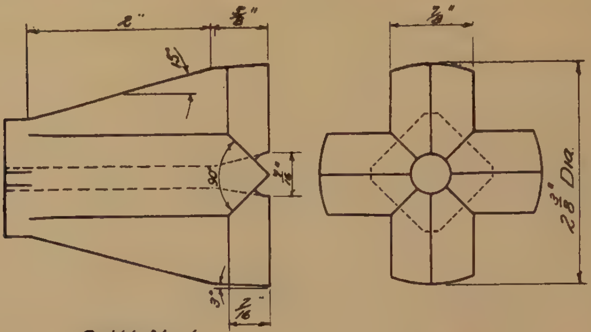
Dr. Donald H. McLaughlin classifies the ore as "consisting of cummingtonite chlorite schist of the Homestake formation with thin local layers of quartz sand and some original iron magnesium carbonate, mineralized with numerous irregular veins, lenses and saddles of white quartz and with ankerite, chlorite, arsenopyrite, pyrrhotite, pyrite, magnetite and gold. . . . Formations cut include the above mentioned Homestake; the Ellison which includes fine-grained gray slate and slaty mica schist, thin black quartzites, and medium-grained gray micaceous quartzites; the Poorman-De Smet, which is coarse-grained garnet mica schist, fine-grained banded greenish chlorite garnet-quartz-mica schist and fine-grained banded gray dolomitic phyllite. The pre-Cambrian rocks and the ore bodies in them are interrupted by dikes of rhyolite porphyry, which occur in a ragged zone about 200 ft. broad." Both ore and country rock may be classified as hard, tough and highly abrasive.

Advantages of the New Gauge

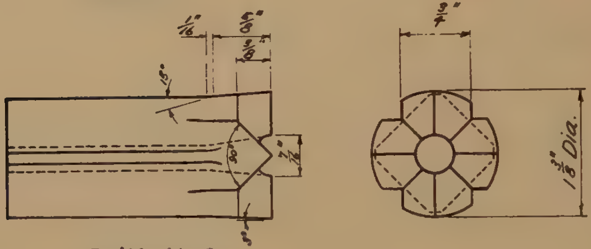
About 4000 tons of ore is milled daily at Homestake and this tonnage is produced by the mine in a 6-day week. During the first 7 months of 1939, all mining, including development, used an average of 3189 pieces of the old standard drill steel per day, but during the succeeding 5 months consumption dropped 18.7 per cent, to 2594 pieces daily for the new gauge. Moreover, bit and shank breakage diminished noticeably, the former by two-thirds and the latter by one-third, undoubtedly because of smoother drilling, which minimized vibration and lessened crystallization due to steel fatigue. Reduction in steel sharpened in turn cuts distribution costs and proportionately increases available stocks of steel.

Over the complete range of lengths, from starters to 10 ft., the average saving in bit cross section is 25 per cent. For a given depth of hole a set of steel will then advance correspondingly faster, and at the same time consume less power and steel per foot.

In a number of tests employing standard brands of smooth-bore hollow 1-in. quarter octagon steel made up with $\frac{1}{8}$-in. gauge changes, a mounted

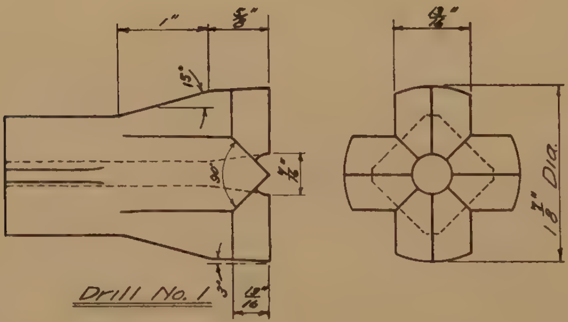


Drill No 1

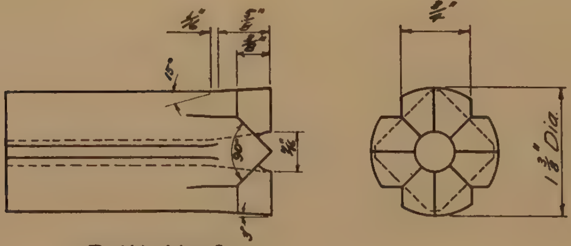


Drill No 9

FIG. 1.—DRILL BITS, ⅛-INCH GAUGE, 1-INCH QUARTER OCTAGON STEEL.



Drill No. 1



Drill No. 9

FIG. 2.—DRILL BITS, ¹⁄₁₆-INCH GAUGE, 1-INCH QUARTER OCTAGON STEEL.

drifter advanced 11.03 in. per min. in medium hard porphyry, whereas with the new 1/16-in. gauge changes 13.78 in. per min. were cut, which is 24.93 per cent faster. This checks the percentage difference in cross section of the bits over the complete range of lengths. In very hard quartzite the increase was from 5.09 to 6.65 in. per min., or 30.64 per cent betterment. In other words, based on identical operating conditions as to mechanical equipment, air pressure, rock texture, machinemen, etc., drilling speed is a function of bit areas. Taking into consideration all operating variables, actual reductions in drilling time may best be expressed as in Table 2. As a case in point, 1½ shifts were required to drill a raise round in hard quartzite with the large bits, but the new standard steel finished the drilling in 7 to 8 hr. Where formerly extra hard faces could not be drilled within the shift, they are now completed in 8 hr. or less.

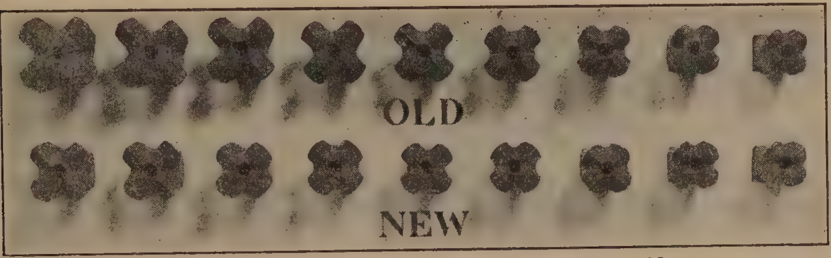


Fig. 3.—Old and new sets of steel from starters to 10-foot lengths.

Table 2.—*Reductions in Drilling Time*

Type of Work	Time Saved, Hr.
Raising	1½ to 2
Drifting and crosscutting	About 1
Shaft stripping	1½ to 2
Timber stopes	25% of actual drilling time
Shrinkage stopes	20% of actual drilling time

By and large, the bit that cuts with minimum gauge loss will make the greatest advance without change. In many places where 12-in. changes were necessary with the large bits, 24-in. changes are now common, and no difficulty is experienced in following. Furthermore, less trouble arises from stuck steel.

Time and trouble are saved in collaring holes. Breakage of starting bits has been measurably reduced. The smaller bit may be used advantageously in blockholing large boulders where the use of a second follower steel is essential.

With the smoother operation of the drills through less vibration, maintenance costs of air drills have been bettered. Consumption of compressed air has been cut 19.5 per cent.

Not the least important has been the lowering of the dust hazard in the machineman's breathing zone. Here again the betterment has been almost exactly proportional to the reduction in bit size—in other words, 20 to 25 per cent. Such an improvement may often mean the difference between a permissible dust count and an excessive one. Betterments of this type deserve serious consideration.

SUMMARY AND CONCLUSIONS

The reduction of drill-steel bit-gauge changes from ⅛ to 1/16 in. has been highly beneficial both directly and indirectly through the reduction of bit and shank breakage, steel and compressed-air consumption, rock-drill maintenance, distribution expense, drilling time, number of changes necessary to drill a hole, collaring time, stuck steel and generation of dust. Drilling is a smoother operation because of less vibration. Several thousands of dollars a year are saved in drill-sharpening expense and much more in compressed air and in the savings attained in materially lower unit steel consumption. The organization is unanimously in favor of the new bit gauges.

The success of the innovation suggests future experiments with ⅞-in. steel and slightly smaller bits than are now employed, continuing, however, with 1/16-in. changes. This will introduce new problems, such as modification of the chuck end of the machine, the ability of the steel with smaller cross section to withstand the added work it must do to equal the 1-in. steel, and whether sufficient powder can be loaded into the hole. Proposed sizes for such experiments would be starters with 1¾-in. bits down to 10-ft. steels with bits of 1¼-in. diameter.

ACKNOWLEDGMENTS

The help and friendly criticism of those of the Homestake organization who aided in the compilation of the data for this paper are gratefully acknowledged.

Air Transport at Agua Fria

By Felix B. Shay,* Member A.I.M.E.

(New York Meeting, February 1940)

The Agua Fria mine is in eastern Honduras at Lat. 140° 06′ N., Long. 86° 36′ W. Danli, the seat of the local municipal government, is the nearest town. The property is approximately 44 miles east of Tegucigalpa, the terminus of a truck road from the Pacific coast. The mine lies at an altitude of 3300 ft. with surrounding elevations as high as 6000 feet.

For 20 years the oxidized croppings of the Agua Fria vein had been worked intermittently when in 1927 an attempt was made to open the property. A shaft was sunk 300 ft. and considerable ore was blocked out, but before a mill could be installed financial difficulties arose and the mine reverted to the original owners. Erle P. Halliburton, of Los Angeles, purchased the property in 1934. At that time a careful consideration of all factors involved showed that air transport was the only practical method of moving the equipment and machinery necessary to commence production. The other alternative was by muleback, which required sectionalizing all equipment to a maximum load of 200 lb., and during the rainy season this type of freighting was subject to serious delays.

It was decided to purchase a trimotor Ford powered by three Pratt and Whitney Wasp B motors developing 400 hp. each. There were several reasons for the choice: (1) this type of plane was available at a reasonable price, owing to its replacement by faster models on most of the airlines in the United States; (2) the ship was capable of carrying a pay load of 3000 lb. over the required distance; (3) since this airplane had been widely used there would be no difficulty in securing competent personnel or in purchasing spare parts.

It was necessary to rebuild the plane for cargo service. The windows were removed and the fuselage was entirely covered. A hatch was cut in the top of the fuselage to receive heavy crane loads and pieces of freight that were too large or too long to go through the rear door. The seats were removed and the interior of the plane was lined with Duralumin sheets. A heavy-duty landing gear was installed and the floor was reinforced.

The nearest site for an airport was two miles north of the mine. The necessary grading was done by a small track-type tractor with a wheeled grader, which was brought in overland. Six weeks was required for bring-

---

Manuscript received at the office of the Institute Nov. 14, 1939. Issued as T.P. 1156 in Mining Technology, January 1940.

* Mining Engineer, Allentown, Pa.

ing this equipment from Tegucigalpa. When the grading of the airport was finished, the plane brought in a truck and the tools to construct a road to the mine. At this time the plane was based at San Lorenzo, on the Pacific coast, a flying distance of 90 miles, and operated from that point during the preliminary construction period. After production started, in August 1935, it was decided to fly the freight to Tegucigalpa and truck it

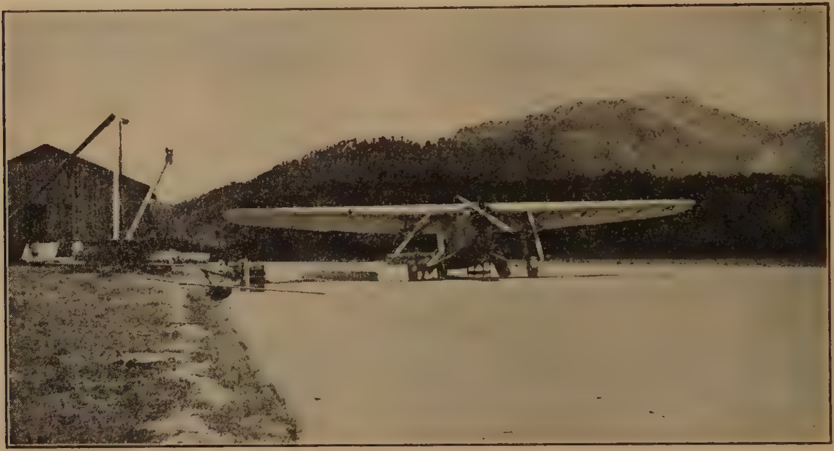


FIG. 1.—AIRPORT AT AGUA FRIA.

TABLE 1.—*Costs*

Item	December 1936	Year 1936
Total pay load hauled, lb.	196,601	1,989,388
Number round trips	34	391
Average pay load, lb.	2,891	2,544
Total flying hours	40.42	423.17
Gallons gasoline per hour	67.55	71.01
Quarts oil per hour	5.54	6.43
Operating expense, per hour:		
Gas and oil	$21.47	$22.94
Pay roll	23.89	24.61
Radio	1.73	2.13
Motor overhaul	9.37	9.37
Other repairs	6.34	5.13
Airport	1.88	6.81
Agent and ground crew	5.40	5.00
Insurance and taxes	1.84	2.29
Total cost per hour	$71.92	$78.28
Cost per ton	$29.60	$33.33
Cost per ton-hour	49.75	61.60
Cost per ton-mile	0.64	0.72

to and from the coast, therefore a T-shaped hangar was constructed at Agua Fria and the plane was based there (Fig. 1).

The flying personnel consisted of a pilot and a co-pilot who also acted as mechanic, making the necessary repairs and checks. A radio operator at Agua Fria and an agent at Tegucigalpa formed the ground crew. No radio was installed in the plane but communication was maintained with the airport in Tegucigalpa and no flights were attempted in bad weather. All loadings were carefully double-checked and an ample reserve of gasoline was always carried. Motors were changed on a regular schedule and sent to the United States for major overhaul.

A successful operation of this type depends in large measure on regularity of flights. This in turn depends almost entirely on the capability and dependability of the pilot and mechanic. The necessity for competent and experienced men cannot be too highly emphasized. The record of 3,500,000 lb. flown in 28 months without a mishap or loss of time due to repairs is ample tribute to the personnel involved.

Table 1 gives the cost of operation for the month of December 1936 and for the year 1936. For two months in 1936 the mill was shut down, which increased costs for that year.

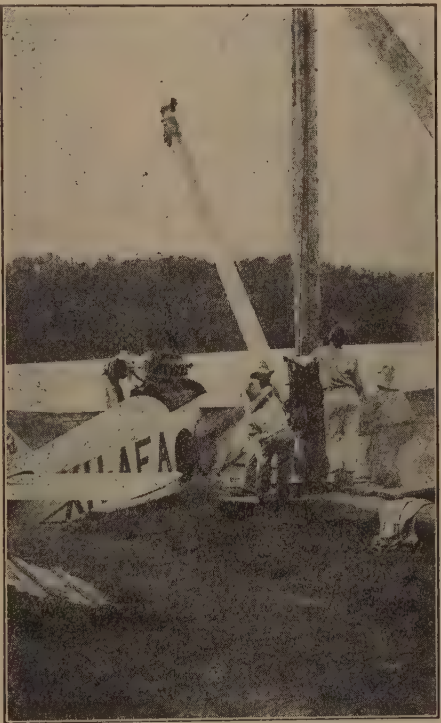


FIG. 2.—UNLOADING SYMONS CONE CRUSHER.

An analysis of these costs shows that it is important to load the plane as heavily as is compatible with good safety practice, therefore the unbalance of loads on a round trip—flying one way with a light load—was the greatest disadvantage. Every effort should be made to schedule the arrival of supplies in order to maintain an even flow of freight. It is also worth while to keep a close check on the ratio of concentration in the mill, as a much higher recovery can be made with an increased tonnage of concentrate, which will effectively balance the incoming loads.

As the performance of a single unit is never as efficient as a multiple operation, it was decided in May 1937 to make a contract with a local aviation company for all freight hauling. The aviation company operates a large fleet of planes, which were already flying several runs a week in the vicinity of the mine, so that it was possible to effect a substantial reduction in cost. The contract was let on a straight poundage basis, no minimum load being specified. This service was operated through 1937 and 1938.

Early in 1938, the life of the mine having been substantially increased, it was decided to build a truck road about 33 miles long to connect with the nearest road into Tegucigalpa. After the road was completed, in December 1938, all freight was moved by truck at a considerable saving over the cost of air transport.

It is apparent that the airplane can compete with mule and ox transport over rough country and that the advantages of load, rapidity, assurance of delivery and flexibility give the plane a definite superiority. It is also evident that an air transport company can fly freight more economically than a single operator. However, the airplane cannot compete with other forms of mechanical transport where facilities are available or can be constructed at a cost of less than the estimated saving over air transport.

DISCUSSION

(*W. E. D. Stokes, Jr., presiding*)

W. E. D. STOKES, JR.,* New York, N. Y.—I have secured the following data from Canada, which I submit as discussion for Mr. Shay's paper:

Water-borne transport is generally the cheapest of all. In the Canadian "bush" costs per ton-mile vary from 8¢ to 20¢, depending upon distances and number and length of portages. Suitable roads for truck haul, with graveled surface from 10 to 12 ft. wide, are built for between $1800 and $2500 a mile, depending upon the amount of rock work and bridges involved. Maintenance costs are from $50 to $100 a mile annually. Winter roads are cleared for 25 ft. in width of stumps and stones at a cost of $50 to $750 a mile. When about a foot of snow has fallen tractor rollers compact it for a width of 10 to 12 ft. and 500-cu.-ft. water tanks mounted on sleighs will "ice" about 1/8 mile. Maintenance and icing costs from $300 to $350 per mile. Tractor hauling runs about 10¢ to 30¢ per ton-mile exclusive of road preparing and depreciation but including repairs, overhead, insurance and supervision. Truck haul is three times as rapid as tractor haul; the investment in equipment will be less while the cost of the road is likely to be heavier; the over-all cost on a contract basis is between 30¢ and 45¢ a ton-mile.

Mr. C. A. Banks, Managing Director, Bulolo Gold Dredging Ltd., has furnished some current figures on aerial freighting, as follows: "From Nov. 30, 1936 to May 31, 1939, we have carried an average of 420 short tons monthly, and the cost per ton has amounted to $34.76 or, including amortization at $7.60 per ton, a total cost of $42.36. These figures are made up as shown in Table 2."

The figure of $42.36 per ton at Bulolo for aerial freighting a load distance of 35 miles works out to the over-all cost of $1.21 per ton-mile. There are a number of instances in Canada where on longer and more economical hauls complete mining plants have been moved by air at prices of from 80¢ to $1.00 per ton-mile.

Aerial freighting costs will have to be substantially lowered in the future to compete with other means of transportation. A more indestructible type of plane would

---

* President, Onward Construction Co.

TABLE 2.—*Aerial Freighting at Bulolo*[a]

Item	Cost per Ton of 2000 Lb.	Percentage of Total
Loading and unloading	$ 0.70	2.01
Operating costs	16.67	47.96
Maintenance and repairs of machines	5.73	16.48
Maintenance of airdromes and building	1.00	2.88
Management	5.93	17.06
Insurance	4.73	13.61
Total	$34.76	100.00
Amortization	7.60	
Grand total	$42.36	

[a] Figures from Mr. Banks.

cut depreciation and maintenance costs. Motors requiring less attention would cut operating costs. Insurance rates are now very high and should be reduced.

It should be borne in mind that all these examples of airplane freighting for mining involve short hauls and slow planes. A better record can be achieved under more nearly ideal conditions, as, for example, air lines operating in the United States. United Air Lines Douglas DC-3 planes carry 3 tons at 180 miles per hour a distance of 1000 miles a day and 200 days a year at a cost of $122,400. This works out to 34¢ per mile. These planes cost $80,000 new and are written off the books in four years, after which they sell second hand for $30,000 and can be used for another 6 to 10 years.

Here is one more example. The newest and most efficient load-carrying plane is deemed to be the new Martin bomber No. 167. It has been estimated by Grover Loening, naval aircraft designer and director of Pan American Airways, that the Martin No. 167 could operate as a freight carrier on a 2000-hr. a year basis for $125,000, carrying 3 tons at 300 miles per hour. This works out to 28¢ a mile flown or 9⅓¢ a ton-mile. Mr. Loening thinks that if the traffic warranted the Martin No. 167 could be operated between two points 1200 miles apart, such as Miami and New York, at the rate of two round trips a day for 240 days a year. This would, under optimum conditions, work out to cost under 5¢ per ton-mile.

Hazards from Gases in Metal Mines and Protections against Them

By E. H. Denny,* Member A.I.M.E.

(Tucson Meeting, November, 1938)

In the past few years many men, including technically trained engineers, have been asphyxiated in metal mines, coal mines, tunnels and surface wells where a few relatively simple precautions with regard to ventilation and air testing would have saved them.

Death from mine gases is analogous in some respects to death from poison and from drowning. The respiratory passages of a victim who has breathed oxides of nitrogen from burning explosives are badly irritated and a few hours or even a day after exposure he may develop a quick and fatal pneumonia; if, however, he enters without respiratory protective apparatus an atmosphere deficient in oxygen (a condition that may occur in unventilated or poorly ventilated parts of all underground mines) he collapses and if not rescued dies more quickly and surely than a lone nonswimmer who falls suddenly into deep water. An example of an atmosphere virtually without oxygen was demonstrated by an air sample taken by a Bureau of Mines engineer wearing oxygen breathing apparatus in a Cripple Creek shaft some years ago, during the recovery of the bodies of two victims of asphyxiation; analysis of the sample by the Pittsburgh laboratory of the Bureau showed the presence of 18.40 per cent of carbon dioxide, 0.27 per cent of oxygen, and 81.33 per cent of nitrogen, although the atmosphere apparently was clear and normal. In a mine one can pass from good, breathable air to an atmosphere that will kill as quickly as one can dive from a board to deep water below.

SOME INSTANCES OF ASPHYXIATION BY GASES IN METAL MINES

The following typical examples of comparatively recent instances of asphyxiation in metal mines were taken chiefly from reports of various kinds of the U. S. Bureau of Mines.

Manuscript received at the office of the Institute July 25, 1938. Published by permission of the Director, U. S. Bureau of Mines. Issued as T.P. 984 in Mining Technology, November 1938.

* District Engineer, Health and Safety Branch, U. S. Bureau of Mines, Denver, Colorado.

Irrespirable Gases

A mine foreman entered a drift that was turned off a long metal-mine tunnel, in the Cripple Creek district, Colorado. He was alone and using a flashlight. All operations along the tunnel had been suspended a day or two before because irrespirable gases were issuing from the strata. This drift had a door at the entrance and a pressure fan forcing air into the mine workings to hold back rock gases and provide respirable air to the workers. An extinguished carbide lamp near the open door, the foreman's body some distance beyond the door in the drift with a flashlight near by, and a fan that had not been operated seemed to tell the story, despite the absence of witnesses. The foreman, without means of gas detection or protection, entered a drift in which he should have suspected the presence of irrespirable gases. Perhaps many times before he had entered atmospheres in which a carbide lamp would not burn and had "gotten away with it," but this time he did not.

In some formations in the Cripple Creek district an irrespirable mixture of gases, consisting mainly of varying amounts of carbon dioxide and nitrogen, issues from the strata and from crevices in the strata, particularly at times of low barometric pressure. This gas mixture may contain enough carbon dioxide to lie near the bottom of drifts and in winzes, or it may contain only a small amount of carbon dioxide with the remainder mainly nitrogen and may lie in a raise. Tests by the Bureau of Mines indicate that these rock-strata gases may be formed by the interaction of water and oxygen on various constituents of certain rock formations, with liberation finally of carbon dioxide gas and absorption of oxygen, leaving the residual nitrogen.[1] Irrespirable gases consisting mainly of nitrogen and carbon dioxide are found also in the Tintic district, Utah, and at other places. Frequently they occur in unventilated drifts, winzes, and raises in mines, and may be expected in mines producing gold, silver, copper, lead, zinc, iron, or almost any other metal or mineral. Surface wells frequently have gases of this description, and numerous tragedies have resulted therefrom.

Two men were repairing and timbering a single-compartment shaft 180 ft. deep, in a metal mine in California, which had been closed for years. Bad air had been noticed in the shaft, and an attempt was made to establish ventilation by a stove placed on the surface and connected down the shaft by a 4-in. pipe. The two men failed to return to the surface one evening, and a rescue party of three went to the shaft and climbed down the ladderway, using lighted candles for testing the air. At a depth of 150 ft. the candles were extinguished by the air. Using a flashlight one of the men went on down to the bottom of the shaft. He found the two men still living but unconscious and started back up the ladder. After climbing about 30 ft., he was overcome by the gases and fell, breaking his neck. The skip was operated, the compressed-air line opened, and ventilation by the stove attempted in an effort to stir up the heavy gases. Then three men wearing gas masks attempted the rescue but were overcome; they did not realize that gas masks furnish no oxygen and therefore provide no protection against oxygen-deficient atmospheres. After some hours enough fresh air was mixed with the bad air to allow recovery of the bodies.

---

[1] References are at the end of the paper.

Two men using electric cap lamps were lowered in a bucket down an unventilated winze in a metal mine in Nevada. Apparently they were proceeding to a nonworking and unventilated level for reasons of their own and on their own responsibility. They were overcome and fell down the winze. The man at the hoist then started down the winze manway, but also collapsed and fell. In an effort to save these three men, three more men died. Finally the six bodies were recovered, four of them by the use of breathing apparatus.

Gases Produced by Blasting

An estimated quantity of 170 sticks of 40 per cent gelatin dynamite had been fired in two short drift faces off a sublevel winze in a metal mine in New Mexico. This winze was 125 ft. below the level and was ventilated by compressed air. Ordinarily the compressed-air line was left open at blasting time, but on this occasion it had been closed. Eight hours later three men came to the collar of the winze; one of them climbed down 50 ft. and connected the air hose to the closed valve and opened the valve. After air had blown for 45 min. one man started to descend the ladder and fell after climbing down about 60 ft. The other two men probably did not realize that the man had been overcome by asphyxiating gases and they also were overcome in attempts to rescue him. Thirty or forty minutes later a man with a rope tied around him climbed down the winze and was able to tie ropes around the bodies. In this instance the oxygen in the atmosphere probably had been so diluted by mixture with the gases evolved from the explosives used in blasting that it would not sustain life. Considerable carbon monoxide was produced by the blasting, and it is possible also that heavy gases may have issued from the strata and mixed with the gases from the blasting. The quantity of air that was issuing from the compressed-air line is not known, but it was probably about that used by an ordinary drill, say 75 to 150 cu. ft. of free air per minute. In 45 min. the quantity of air thus released would be small compared with the volume of irrespirable gases already present. Also, it is known from records of similar accidents, that a compressed-air jet may be very slow in moving heavy gases and may merely cut holes, as it were, in the gas blanket without moving it appreciably.

About 340 sticks of 40 per cent ammonia gelatin had been fired with fuse in three drifts and two raises in a mine in the Lake Superior district. Following the blasting it was discovered that a miner was missing. He was found groping his way out of his working place. He had not been injured by the blasting but was not feeling well. He was sent outside and examined by the doctor, who found him apparently about normal but advised him to go to the hospital for observation; instead he went to the boarding house, and at 9 o'clock in the evening, while resting comfortably on the porch, he collapsed suddenly and died at 10 p.m. It is probable that the inhalation of oxides of nitrogen caused death in this instance. The effects of nitrogen peroxide on the respiratory passages are not usually manifest until several hours after exposure; then swelling takes place and the irritation may be followed by an acute attack of bronchitis or pneumonia, frequently with fatal results.[2] Although most of the known cases of death from oxides of nitrogen in connection with explosives have been due to the burning of explosives, research by the Bureau of Mines and others has revealed that when some explosives are detonated oxides of nitrogen may be present immediately after blasting in sufficient quantity to cause serious illness or death.[3]

Blasting in wet ground may produce 5 to 10 times as great a quantity of oxides of nitrogen as blasting under dry conditions. Oxides of nitrogen are quickly diluted by ventilation or even by water absorption, therefore if readily available precautions in ventilation and use of water sprays are taken in entering places after blasting there should be relatively little danger from this source.

The danger of exposure to small quantities of oxides of nitrogen may be judged by the fact that 0.07 per cent is likely to be fatal if breathed for 30 min., or possibly less, and 0.01 per cent to cause dangerous illness if breathed ½ to 1 hr. The Bureau of Mines found[3] in field tests on 60 per cent gelatin that in an unventilated place 10 min. after blasting there were likely to be small but definitely dangerous amounts of oxides of nitrogen that required dilution by ventilation and larger and equally dangerous amounts of carbon monoxide.

Cause Not Identified

Two miners were mucking near the foot of an underground shaft in Colorado, 60 ft. below a drift turned off the shaft. They were loading old muck that had fallen down the shaft over a period of years. Thirty-two holes had been shot in two small stopes off the short drift above the shaft bottom 2 hr. before the miners started to work. Natural ventilation and compressed air were relied upon for ventilation, and had proved adequate. A compressed-air line extended fairly close to where the men were working, but it had never been used because the air in the shaft had been good at all times. After blasting, the stopes were blown out with compressed air. Evidently much of the smoke from the blasting passed down the shaft to where the two miners were mucking, because when one of them went to the hoist station he remarked to the hoistman that it was very smoky below. After rock had been hoisted in a bucket for 4 hr., mucking below suddenly stopped and the hoisting engineer became alarmed. About 2 hr. later mine officials, who went to the bottom of the shaft, found the air good, but the two miners were lying dead with evidence that their carbide lights had burned their clothing and bodies before being extinguished.

A coroner's jury attributed the deaths to carbon monoxide poisoning with the source of carbon monoxide not fixed. This finding was based on the testimony of medical men who examined the bodies, although no blood tests were made. It seems probable that at the time of this accident (the fall of the year) natural ventilation had a seasonal tendency to change and that probably there was a period of definitely stagnant air for some hours after midnight. Irrespirable gases may have entered from the strata, but none of these had been found in the mine before or afterward or during the recovery of the bodies, and it is evident that there was enough oxygen in the air to support the flame of the carbide lamps for a time after the men collapsed. Again, oxides of nitrogen and carbon monoxide from blasting are under suspicion, possibly supplemented by bad air from the strata. Incidentally, it is said that poisoning from oxides of nitrogen gives the same type of red discoloration of parts of the body as does poisoning by carbon monoxide; the fact that such coloration existed had been pointed out by the examining doctors as evidence of carbon monoxide.

Exhaust from Gasoline Engine

Two men were reopening and retimbering a metal-mine shaft in California. They were unwatering the shaft by using a pump driven by a gasoline engine mounted on skids. The pump was lowered as the water receded. An attempt had been made to conduct the fumes from the engine to the surface by 1¼-in. pipe, and tape had been used on the pipe connection to reduce the leakage of exhaust fumes into the shaft. Usually the men descended the shaft to start the engine and then climbed out until the pumping was completed; on this occasion, when they did not appear for lunch the wife of one of them went to the shaft, found the engine still running, and stopped it by pulling a wire provided at the shaft collar for the purpose. A rescue crew was called. Fortunately the air cleared during the time that elapsed before the crew appeared, as they conducted their rescue with a type of gas mask that gives no protection against carbon monoxide, the dangerous gas present in any gasoline-engine exhaust. The two bodies were only 40 ft. below the shaft collar.

Despite the hundreds of cases of persons being overcome and killed by automobile exhaust in garages and similar places, there are far too many attempts to use a gasoline engine in a shaft or in a mine, or so located in proximity to a mine opening that the exhaust fumes may get into the mine. Use of gasoline-powered equipment in or adjacent to mine workings is hazardous in the extreme, as it offers a bad fire as well as asphyxiation hazard, even where apparently good ventilation is normally in effect.

Hydrogen Sulphide

The Bureau of Mines has record of several instances of the presence of hydrogen sulphide in dangerous amounts in underground operations, chiefly in tunnel and caisson work, particularly in sands and sediments. One instance in which the gas was encountered was in caisson work, where workers were sinking in sand under about 20 lb. of air pressure. Hydrogen sulphide appeared first in quantities sufficient to irritate the eyes, and airtight goggles were provided for the workers; however, when the gas is present in quantities large enough to irritate the eyes, it will also cause irritation of the respiratory passages, and in slightly larger quantities is likely to cause collapse and death of workers. In this instance, one worker collapsed, was unconscious for 48 hr. and did not fully recover for some months. Increase of compressed-air pressure, to hold the hydrogen sulphide gas back in the strata, and better airing of the shaft were adopted, and no further trouble was encountered on this job.

It is fortunate that this gas is not commonly encountered in metal mines. It may be produced by the burning and in some instances by the detonation of explosives containing sulphur or the use of dynamite in blasting sulphide ores. It is found in extremely toxic amounts not only in blasting some sulphide ores but also in natural gas and oil in certain fields. It is one of the most poisonous gases known; exposure to as little as 0.01 per cent by volume in air may cause death, and 0.1 per cent will cause serious symptoms in man within a few minutes.

Hydrogen sulphide gas is readily detectable[2] in very small amounts by its characteristic rotten-egg odor; in dangerous amounts, however, its presence is detectable only on the first inhalation or two, after which the respiratory centers are paralyzed.

DETECTION OF AND PROTECTION AGAINST GASES IN METAL MINES

Oxygen-deficient Air

The asphyxiations cited as caused by nitrogen and carbon dioxide may be classified as accidents due to oxygen-deficient air. Carbon dioxide gas does not support animal life. It is not poisonous and in small percentages in ordinary air stimulates breathing. It is formed by the breathing of man and animals, by the oxidation and decay of timber, and by the action of acid water on carbonates; it is also one of the gaseous products of blasting. Nitrogen forms approximately 79 per cent by volume of normal air. When the oxygen of the air is burned or combines chemically with mine minerals and timbers, the nitrogen, being an inert gas, is left unchanged. A man may be suffocated through lack of oxygen by both nitrogen and carbon dioxide, as in drowning.

The presence of oxygen-deficient air may be detected by its extinguishing effect on a candle, match, flame safety lamp, or carbide lamp. The first three are extinguished in air containing enough oxygen to support life; usually a carbide lamp gives warning in time for a man to withdraw, but it may not be extinguished until the oxygen content in the air falls from the normal 21 per cent to between 12 and 13 per cent, and instances are on record of loss of life by asphyxiation of men who depended on the burning carbide lamp for safety.

The oxygen deficiency at which men will collapse seems to vary with the individual and with the amount of his exertion, and possibly is influenced by other conditions also; an extreme example of this is shown by a Bureau of Mines test in which a man sitting and inhaling and exhaling from a bag containing air and stick potassium hydroxide to absorb the exhaled carbon dioxide did not become unconscious until the oxygen content of the air reached 6.01 per cent.[4] Usually collapse from oxygen deficiency is sudden, with no warning symptoms. The Bureau of Mines considers[5] that any mine atmosphere normally worked in by man should contain at least 19 per cent oxygen and not more than 1 per cent carbon dioxide.

Carbon monoxide gas is slightly lighter than air, odorless, and difficult to combine with most substances except through burning. It is evolved from blasting, from practically any type of explosive, from gasoline-engine exhaust, and from underground fires, particularly timber fires. It does not occur except through incomplete combustion of carbon-containing material and its presence may be suspected wherever the products of combustion are present. It may be oxidized to the relatively harmless carbon dioxide, as in the All-Service gas mask, through the catalytic action of manganese dioxide and copper oxide. It has the very dangerous property to man of combining with the hemoglobin of the blood to form a compound that prevents the blood from absorbing oxygen from the air. The affinity of carbon monoxide for hemoglobin is about 300 times that of oxygen; therefore, when only a small amount of the toxic gas is in the air breathed much of the hemoglobin is combined with it and cannot carry oxygen to the tissues. The action, however, is reversible in the presence of oxygen, and when a person thus poisoned is resting in fresh air the hemoglobin gives up the carbon monoxide and is again able to absorb and carry oxygen to the tissues. Breathing small amounts causes headache and nausea; 0.2 per cent or more quickly results in unconsciousness and may result in death.

Carbon monoxide has no effect on the flame of a lamp or candle in small percentages that may be immediately harmful to man. It may be detected by a carbon monoxide detector or by a canary. The writer has had little confidence in the use of canaries since experience with a bird that proved more resistant to the gas than the rescue crew accompanying it. The carbon monoxide detector[2] depends in principle upon the fact that carbon monoxide when passed through iodine pentoxide releases free iodine and imparts a greenish color to the granules in the detector tube. Precautions in the use of the detector include seeing that all joints are tight and that the charcoal used to absorb interfering gases is fresh. This instrument is very useful in proving the presence or absence of small amounts of carbon monoxide in smoke issuing from mine fires and in mine atmospheres following coal-dust or methane-gas explosions; however, since its action is not automatic a person carrying a carbon monoxide detector may easily enter a noxious atmosphere and be overcome without having tried to use the detector. The canary acts automatically and in this has a definite advantage over any mechanical portable detector yet devised.

Methane

Examples are not given of injury or death of metal miners from the explosive gas, methane, occasionally found in metal-mining operations. Methane has been found in mercury mines in California, in metal mines in the Lake Superior district, and in dewatering old shafts in Colorado, Montana, and other states. Usually its presence is known only when someone explodes it with a lighted carbide lamp. Its physiological effect on man is virtually the same as that of nitrogen. Instances are known of asphyxiation of men by methane in coal mines, but the writer knows of no such instances in metal mines.

Obviously no type of open-flame light is suitable for testing air containing explosive gases. Methane is commonly detected in coal mining and in salt mining with the flame safety lamp, but the use of this lamp is not generally known in metal mines. Misuse of the flame safety lamp, such as incorrect assembly of the numerous parts of the lamp or disassembly of the lamp in explosive gas, has caused several very serious disasters in coal mining, and anyone that uses it should be thoroughly familiar with it and with its limitations. There are several types of combustible gas detectors on the market which, in the experience of engineers of the Bureau of Mines, give satisfactory results in the detection of quantities of methane ranging from 0.5 per cent upward, and these devices seem to be reasonably foolproof, though usually they give no warning of oxygen deficiency.

Ventilation and Equipment Necessary

The accidents discussed and numerous others of which the Bureau of Mines has information disclose that lack of ventilation and failure to have or to use proper equipment were the principal causes of accidents, many of them fatalities. In some instances men have lost their lives through ignorance while in others experienced officials have lost their lives through foolhardiness. Experienced mining men often have the utmost confidence in the ability of their lungs and hearts to withstand exposure to gas or smoke without the modern aids of ventilation or protective devices—in other words, they believe that they are "smoke eaters." They may take many chances and sometimes may survive, but the mining industry as a whole inevitably suffers an unnecessary toll of human life from such chance-taking, and ultimately the chance-taker himself is very likely to take one too many chances. The large metal-mining operations of Arizona have mechanical ventilation induced by large main fans supplemented by auxiliary ventilation. Even in these mines it is well to bear in mind that hazards from gases may exist in sections that are not well ventilated. With auxiliary fans or blowers the ventilation afforded at dead ends or in stopes may be good, indifferent, or bad, depending on the capacity of the blower and suitable extension of pipe or tubing, and, equally important, the placing of the fan intake in a source of fresh air of adequate volume. The principal advantage of a blower, if it is not set up in a fresh air current, may be simply to give the worker moving air and absorb perspiration, and there may be much recirculation of air and gases produced by mining operations. Many small metal-mining operations in the United States still depend on natural ventilation, supplemented by compressed-air lines, and in such mines opportunity for asphyxiation of men is likely to occur at almost any time.

The question frequently arises as to proper protective equipment against gases; unquestionably the best protective equipment is good ventilation. However, various kinds of equipment giving respiratory protection are available. The "All-Service" gas mask gives emergency protection in atmospheres containing limited percentages of carbon monoxide, oxides of nitrogen, hydrogen sulphide, and a large number of industrial gases; protection is given however only if there is enough oxygen in the atmosphere to sustain life and the total poisonous gases present do not exceed 2 per cent by volume. The death of two miners wearing these masks in an oxygen-deficient atmosphere in a mine near Gallup, N. M., in the fall of 1937 is an example of the limitation in the use of this mask. Masks supplying fresh air through hose give complete protection for short distances in irrespirable atmospheres. The self-contained oxygen breathing apparatus gives complete protection if in good order and properly adjusted to a wearer trained in its use. The Bureau of Mines recommends[6] that the maximum distance of travel in irrespirable air be 1000 ft. and that five men constitute a crew, with five more men with apparatus as a reserve crew; yet every year the writer has one or more requests for the loan of apparatus to use in examining a vein a mile or more inside an old mine filled with gases that extinguish a carbide lamp. If such a lone explorer would consider that he is in much the same position as a deep-sea diver without any line connecting him to safety, and with no chance of rescue in event of trouble, perhaps fewer requests of this kind would be made.

Although this paper reiterates facts known to most mining men, it is hoped that it will be effective in impressing the necessity of some rather simple precautions in the testing of air and ventilation and that thereby some needless loss of life may be prevented.

REFERENCES

1. E. H. Denny, K. L. Marshall, A. C. Fieldner, A. H. Emery, W. P. Yant and W. A. Selvig: Rock-strata Gases of the Cripple Creek District, Colo., and Their Effect on Mining. U. S. Bur. Mines *Bull.* 317 (1930). 66 pp.
2. J. J. Forbes and G. W. Grove: Mines Gases and Methods for Their Detection. U. S. Bur. Mines *Miners' Circ.* 33 (Rev. June 1937). 82 pp.
3. G. St. J. Perrott, L. W. Babcock, C. D. Bitting and G. W. Jones: Toxic Gases from 60 Per Cent Gelatin Explosives. U. S. Bur. Mines *Tech. Paper* 482 (1930). 30 pp.
4. Y. Henderson and J. W. Paul: Oxygen Mine Rescue Apparatus and Physiological Effects on Users. U. S. Bur. Mines *Tech. Paper* 82 (1917). 102 pp.
5. Mines Safety Board: Recommendations of the U. S. Bureau of Mines on Certain Questions of Safety as of Oct. 1, 1936. U. S. Bur. Mines *Inf. Circ.* 6946 (1937). 45 pp.
6. Mine Rescue Standards; a Tentative Study Prepared by a Committee Appointed at the International Mine-rescue Standardization Conference, September 1921. U. S. Bur. Mines (1923). 44 pp.

DISCUSSION

(P. G. Beckett presiding)

S. H. LORAIN,* Moscow, Idaho.—It seems to me that the dangers of long continued breathing of small quantities of poisonous gases should be more strongly emphasized. Although, as Mr. Denny points out, many underground men are still careless at times, most of us are continually alert to the dangers of strong concentrations of gases, such as occur in the immediate vicinity of recent blasting, but we are much less likely to be on our guard against minute concentrations seeping through the workings for relatively long periods. These may be just as deadly, and are much more insidious than the strong concentrations near the face. They are particularly likely to occur where mechanical ventilation is not used. Some years ago, at a northern copper mine, a large part of the shift was overcome repeatedly, wholly or partly, as a result of permeation of the entire workings with dilute gases. This, of course, represents a considerable direct economic loss as well as a menace to health.

---

* Mining Engineer, Metal Mining Methods Section, Mining Division, Bureau of Mines.

Another example is that of two miners working, on an apparently well ventilated level, at the top of a raise. The raise was partly full of muck and therefore considered to be sealed. During the shift there was considerable blasting on the lower level near the bottom of the raise. At the end of the shift the two men were dead, or died shortly afterward. Evidently enough gas had seeped through the muck in the raise to be fatal when breathed for a long period. Such accidents may very easily happen when there is much blasting on shift.

Safety and Health Efforts of the Anaconda Company at Butte

By John L. Boardman*

(New York Meeting, February, 1938)

The Anaconda company has never indulged in any employee activities at Butte which might be termed paternalistic, but it has exerted a vast amount of effort in care of its employees during working hours. The reason for this policy is not lack of interest or that the company does not feel the effect of employees' activities while they are "off the job"; it is that the company and the city of Butte have grown up together and each has been adequate in its own sphere, the company in providing employment at the highest possible wage and the city in providing housing, sanitation, commercial services, medical services, schools, churches, amusements and entertainments.

The one exception to this independence between the company and the community is the Butte Water Co., which is a wholly owned subsidiary of the Anaconda Copper Mining Co. Mention of this relationship is made here because the water supply of such a city as Butte is one of the principal factors in determining the health of the people. Very early in the development of the Anaconda company an adequate and dependable water supply for the operations became necessary, therefore the company took over this important community function.

So well has this activity worked out that during the whole period of time covered, in which the company has expanded to become practically the sole mine operator and the community has grown from a small mining camp to the status of a modern city, there has been no shortage of water or any outbreak of water-borne, epidemic disease.

Conditions Leading to Organized Safety Effort

A brief sketch of conditions in Butte prior to 1912 is necessary to an understanding of the reasons for placing accident prevention on an organized basis. As early as the year 1884 there were a large number of small independent mining operations depending principally upon the production of silver, although even then copper was becoming prominent in the value of total production for the district. The Butte camp was then just emerging from the log cabin stage to that of a modern city.

Manuscript received at the office of the Institute Jan. 22, 1938; revised June 27, 1938. Issued as T.P. 993 in Mining Technology, November 1938.

* Chairman, Bureau of Safety, Anaconda Copper Mining Co., Butte, Montana. Member of the A.I.M.E. since April 1939.

As an indication of the rapid growth of Butte at that time we find the value of metals produced by Butte mines in 1885 was eleven million dollars and five years later, in 1890, had increased to twenty-five millions.

The first record of any attempt to hospitalize victims of mine accidents is of the establishment of a workingman's hospital by Dr. O. B. Whitford in 1878. He contracted to care for accident cases by having each employee pay him one dollar per month, for which he rendered all necessary medical and surgical treatment for illness and for injuries due to and arising out of the employment. A number of other doctors also maintained offices in the camp, and injury cases were sent to them by whatever means of conveyance was available. In 1881 the Catholic order, Sisters of Charity, established a hospital in Butte, known as the St. James Home. This is now the St. James Hospital and is still operated by the Sisters of Charity. This hospital was followed in 1882 by organization of the Alice Mining Company Employees Hospital Association, which erected a hospital and clubhouse at Walkerville, a suburb north of Butte. In 1887 the Boston and Montana Co., under the leadership of its manager, Captain Thomas Couch, organized an employees hospital association. The company and Captain Couch, by contribution of personal funds, erected a building at Meaderville, about two miles northeast of the town, and donated it to the employees association. In both of these efforts the employees contracted to pay one dollar per month through the company payroll for medical and surgical services. In 1890 Doctors T. J. Murray and I. D. Freund built the Murray and Freund Hospital, now known as the Murray Hospital. This hospital also entered into contracts with various mining companies for the care of their employees on a basis of one dollar per month each. Thus, the "employees hospital contract system" is as old as Butte and has continued in satisfactory force and effect to the present time.

By that time, 1890, Butte was ceasing to be a silver camp and was rapidly becoming a copper-mining district. Then came the strife of the so-called "copper kings," fluctuating metal prices, boundary disputes and other legal battles, in the final adjudication of which consolidations of mining properties began to take place. Then followed a period of peace and prosperity until, as the mines became deeper, requiring greater pumping and ventilation facilities, better smelting methods and selling means, one by one the smaller operators were unable to meet the rising costs and sold their properties to the Anaconda company, which, by reason of its adequate financing and integrated mining system, was able to operate profitably.

Thus, by 1912 we find Anaconda the principal operator and in a position to adopt modern equipment and methods, such as centralized drainage and pumping, and reduced overhead through centralized shops and supplies. But there were no wet drills, very few mine fans, no improved explosives such as we have today. Underground lighting was done by candles, and a few carbon filament lamps using power tapped from the trolley wires. Tramming was done principally by man power or by horses, with a few trolley locomotives. Hoisting engines were run by steam or compressed air and the small 18-in. gauge mine cars were hoisted from the mine for dumping. There were no detachable bits and tons of steel were hoisted daily for sharpening. It was then that Anaconda began its policy of adopting improvements as fast as inventive genius made them available, and it was this policy that introduced the idea of accident prevention, making Anaconda one of the first mine operators to recognize this activity.

Organization of the Bureau of Safety

Anaconda was in the position of the pioneer with regard to accident prevention. There was no experience from which to draw the necessary information. Mr. C. F. Kelley, then vice-president in charge of operations, felt that accident prevention was an essential part of the duties of the mining officials, and he placed Mr. C. W. Goodale, then manager of the Boston and Montana department, in charge of an investigation to devise methods for organized accident-prevention work. Mr. Goodale spent a year in a survey of safety work and in 1914 offered the present form of organization, a description of which may be found in a paper in Volume 68 of the Transactions.[1]

In compliance with Mr. Kelley's idea that safety work is an essential part of the operator's job, the Bureau of Safety was organized to consist of a General Committee composed of the President of the Company, Vice-presidents, General Managers and General Superintendents, as well as Department Heads. Under the General Committee are seven subcommittees: Mines Committee, Anaconda Reduction Works Committee, Great Falls Reduction Department Committee, International Smelting and Refining Company Committee, Coal Department Committee, Lumber Department Committee and Railway Department Committee.

The Mines Safety Committee is composed of the General Superintendent of Mines, his assistants and the mine foremen, as well as heads of the various service departments such as ventilation, mechanical, electrical, geological, engineering, engineering research, etc.

In addition to the members of the operating department, the Bureau of Safety has a chairman, who supervises mine and plant inspection, ambulance service, mine-rescue apparatus and stations, receives and classifies accidental injury reports, supervises first-aid training and first-aid supplies, bulletin boards and safety-poster service, edits and publishes the employees' safety magazine, safety rule books, etc., receives and supervises accident reports and safety efforts of the smelter committees at Anaconda and Great Falls, Mont., and Tooele, Utah. Each mine or plant employs a safety engineer under the direct charge of the superintendent and foreman of the mine or plant. These men report to the chairman of the Bureau of Safety, although their salaries are carried by the mine or smelter account. This is a perfect arrangement for such work because it places the prevention of accidents and improvement of health conditions squarely up to the man directly in charge of operations and then supports him with an inspection and reporting service with which to keep the operation functioning on these subjects. It is, perhaps, an unusual situation to hold the foreman responsible and yet give him authority over his critic, the safety inspector, but it does not work out adversely. In explanation, an accident is possible in any mine or plant. If such an accident should happen, caused by a bad condition that has not been reported by the safety inspector, the chairman and Safety Committee, in reviewing the circumstances, readily detect the failure on the part of the inspector. Therefore, he reports and recommends conscientiously, and the foreman by the same logic is forced to give his inspector's reports every possible consideration, because should such an accident occur and be due to a bad condition that has been reported by the inspector but neglected by the foreman, that official is at once in a bad position. However, everyone in the Anaconda organization recognizes his responsibilities and is sincere in his desire to improve health conditions and prevent accidents. Consequently, the Bureau of Safety has had nothing but the utmost cooperation, and its members, the operating officials, welcome the services of the safety engineers, so that there is no friction.

---

[1] C. W. Goodale and J. L. Boardman: Bureau of Safety of Anaconda Copper Mining Co. *Trans.* A.I.M.E. (1923) **68**, 8.

Compensation Costs

Although the original purpose in organizing for accident prevention was and is altruistic, developments that followed almost immediately put the stamp of finance upon the effort. The Bureau of Safety was officially organized in 1914 and on July 1, 1915, the Montana Workmen's Compensation Law became effective.

The provisions of this law have been revised several times since its passage and each change has added considerably to the compensation cost of injuries, so that at the present time this is one of the prominent items in operating cost. The law now provides a minimum weekly payment of 50 per cent of the weekly wage with a maximum of $15 per week for injured employees having no dependents. The percentage of weekly wage and maximum amount is scaled upward from this minimum for employees having dependents and is 55 per cent and $17 for one dependent, 60 per cent and $18 for two dependents, 62½ per cent and $19 for three dependents, 65 per cent and $20 for four dependents, and 66⅔ per cent and $21 for five or more dependents. Fatal injuries to men with dependents are compensated at the indicated rate and maximum for 400 weeks and accidental permanent total disabilities at the indicated rate and maximum amount for 500 weeks. There is a waiting period of seven days for injured men with dependents, and payment is retroactive to the first day when disability exceeds three weeks. The waiting period for injured employees without dependents is 14 days with payment retroactive to the first day when disability exceeds six weeks. Thus there are tremendous potential costs involved in compensation for accidental injuries.

SAFETY RULES

The first task of the new Bureau of Safety was to formulate rules for the prevention of accidents. This required a knowledge of the causes of accidents, which could be had only through a study of accident reports. The first rule, therefore, was that all accidents, regardless of whether they caused loss of time or not, must be reported. When these reports were received they were classified by causes and worked out in rates per 10,000 shifts to determine the relative importance of the different causes. Having accumulated some information in this manner, it was found that one of the chief causes of fatalities was falls of persons. Therefore a rule was made that all openings underground must be adequately guarded, chutes protected by grizzlies, shaft stations and cages by doors, manways equipped with landings, shaft repair work done from cages or hurricane decks instead of from the old-style portable staging. In this manner each of the accident causes was made the basis of safety rules. The first rule book was published in 1914; it was revised and republished in 1922 and again in 1937. The method by which the 1937 rule book was made is interesting in that it shows the cooperative spirit of the organization. First, the chairman of the Bureau of Safety made an outline of accident causes and filled in all the rules that seemed to him necessary to cover the requirements. These were typed out in such a manner as to leave space for insertion of a number of rules under each heading. The manuscript was then mimeographed and a copy sent to each superintendent, mine foreman and safety engineer with a request that each add any rules applicable to the accident cause and return the manuscript properly signed within six weeks of receipt. When all the manuscripts had been returned to the Bureau of Safety, a master copy was made containing all the rules submitted, except duplications, which were marked to indicate the individuals who had submitted them. It was not unusual to have the same rule submitted by eight or ten different individuals, thus indicating the importance of such a rule. The master copy was mimeographed and resubmitted to the superintendents, foremen and safety engineers, who again added to, corrected, and criticized. When these master copies had been returned, they were reassembled to include all the ideas, and a meeting of superintendents with the chairman of the Bureau of Safety was held and agreement was reached as to the final rules. These were then assembled and submitted to the legal department for corrections of English, compliance with state statutes, etc. The book was then retyped as corrected and submitted to the manager. Receiving his approval, it was then published. A copy of this rule book may be obtained at cost by applying to the Anaconda Copper Mining Co., Bureau of Safety, at Butte, Montana.

Accident Reporting

The system of reporting injuries is similar to that followed originally except that greater promptitude is now required. For each injury a hospital slip is issued in quadruplicate, one copy going to the hospital, one to the Bureau of Safety, one to the Claim Department and one to the mine file. Each morning before going underground the safety engineer reads the hospital slips issued on the previous shift and then interviews the boss in charge, witnesses to the accident, and when possible, the injured employee. He then makes out his accident report, which goes to the Bureau of Safety, Claim Department, assistant general superintendent and mine file. The Bureau of Safety then classifies the accidents by causes, days lost, etc., and makes a monthly report to the manager. Copies of this monthly report are sent to the vice-president, general superintendent, assistant to the manager, superintendents and foremen. Copies of individual accident reports and a copy of the annual report are sent to the United States Bureau of Mines by the Bureau of Safety and each accident is reported to the State Compensation Board by the Claim Department.

The monthly report to the manager is read and discussed at one of the bimonthly meetings of superintendents and foremen and such individual accident reports as reveal bad conditions as well as all fatality reports are also read and discussed at these meetings. Selections are made from accident reports to form the theme of mimeographed safety posters, especially when the report indicates that the accident was due to violation of a safety rule.

Safety Inspection

The safety engineer daily inspects the mine and the surface plant. Sometimes he is accompanied by the assistant general superintendent, foreman, assistant foreman, shift boss or chairman of the Bureau of Safety. At other times he travels alone. He notes any violation of the safety requirements and makes suggestions to the mine foreman on efficiency, prevention of waste, etc., and takes temperature and humidity reading for his report to the ventilation department. He also supervises the installation of blowers, air tubing and ventilation doors, usually having a small crew of men assigned to him for such installations.

The safety engineer's bimonthly inspection reports are addressed to the chairman of the Bureau of Safety and by him presented at the bimonthly meeting of superintendents and foremen. They are then copied and copies sent to the vice-president, general manager, general superintendent, assistant general superintendents and mine foremen.

In addition to the bimonthly meeting of superintendents and foremen, the general superintendent holds a weekly conference with superintendents at which the general planning of the operations is carried on. The Bureau of Safety, Ventilation, Mechanical, Engineering, and other departments submit their ideas to this meeting as desired.

Instruction and Propaganda

In every other industry as well as in mining a very large majority of accidental injuries are due to incompetence, ignorance, lack of foresight, mental preoccupation or stupidity on the part of the injured, and these human traits are much less subject to influence by supervision in mining than they are in other industries because the mining supervisor can spend only about 2 per cent of his time with each of his men. In mining the boss visits each working place and pair of miners twice a day, spending with each about two of the 480 min. in an 8-hr. shift. The remaining 476 min. the miners are "on their own" in so far as working safely is concerned. It is, therefore, very necessary that the work of the bosses be supplemented by continuous propaganda and instruction to keep the idea of safety in everyone's mind. For this purpose the company carries a membership in the National Safety Council and uses its magazines, pamphlets and poster service. It also subscribes to a good commercial service of daily news photographs and bulletins and uses a large number of homemade safety posters. It also publishes an employees' safety magazine, *The Anode*, now in its twenty-second year. *The Anode* carries tabulations of accident rates, shift bosses' honor roll and numerous articles on accident prevention in connection with the operations. Copies of *The Anode* have been mailed regularly to the Engineering Societies Library since January 1915.

Bonus System for Shift Bosses

Although an accident-prevention bonus for foremen had been tried out and discarded at the very inception of organized accident-prevention work, it was felt that perhaps an incorrect basis had been used. Many other operators were using an accident-prevention bonus system for shift bosses for which they claimed much merit. At the beginning of 1935 the company was faced by an influx of comparatively inexperienced men and a means was sought to encourage the shift bosses to exert greater effort in instructing these men along safety lines. Therefore, as an experiment, an entirely new sort of accident-prevention bonus system, based on accident experience but noncompetitive in character, was adopted. The rules for this bonus system are as follows:

1. To qualify a boss must have a crew of at least 400 shifts per month.

2. Having 400 or more shifts, if the boss goes through a month without a lost-time accident he receives a bonus of $1.50. For the second consecutive month without a lost-time accident he receives $3.00, for the third consecutive month without a lost-time accident $4.50. Thus, his bonus is accreted at the rate of $1.50 for each accident-free month until he attains a maximum of $18 per month, at which amount the accretion ceases. This amount is then paid him each consecutive, clear month and does not further increase.

3. Having acquired a clear month, when the boss then has a serious accident (one involving a loss of more than 14 days), $1.50 is added to his previous month's bonus and then one-half is deducted for each such serious accident. For a slight compensable accident (one involving a loss of 8 to 14 days), one-fourth is deducted and for an injury involving a loss of one to seven days, one-eighth is deducted.

4. Bonus accruals are carried forward at the year's end, enabling bosses to maintain their higher bonus payments with only the fact of having accidents to reduce them. The total cost of these bonus payments has been about one thousand dollars per month.

First Aid, Ambulance and Hospitalization

The company has cooperated with the United States Bureau of Mines in first-aid training, with the result that several thousand mine employees have received this valuable instruction during the past 20 years. It holds an annual first-aid contest and insists that all bosses shall have a knowledge of first aid.

First-aid supplies, consisting of mine stretchers, blankets and a portable first aid kit, are kept at each mine. Bosses, motormen, pipemen, tool nippers, fire watchmen and many others are provided with pocket first-aid packets and a small supply of bandages is kept at each underground station. Two automobile ambulances are kept with a driver in constant attendance, so that there is no unnecessary delay in transporting an injured employee to the hospital.

Fire Protection and Fire Fighting

For the past 40 years the Butte mines have never been entirely free from the effects of underground fires. These are caused by electrical equipment, spontaneous ignition, formerly by open-flame lamps, and perhaps in some cases by the careless use of smoking materials.

The company maintains an extensive fire-prevention service. The safety engineers are very alert in detecting and reporting the possible causes of fires, such as defective or damaged electrical equipment, accumulations of rubbish, etc. The mine foremen and bosses are alert on possible fire causes. Each shift boss has a fire watchman, who begins his shift an hour late and remains to patrol the beat after the regular shift leaves. These men carry clocks, which are keyed at designated stations underground.

The mines are all interconnected and escape signs are posted throughout the length of each exit passage, to guide men in escaping a fire danger. Carbon tetrachloride fire extinguishers of the pressure type are provided at all underground shaft stations, motor-generator sets, large fan installations and other places where there is a possible extra fire hazard. Extinguishers are inspected regularly by the safety engineers and bosses. A means is provided for turning water into the air lines, so that extra water for fighting an underground fire is almost immediately available throughout the mines. Extra hose and nozzles to fit mine air-line connections are kept at each mine. Concrete fire-door frames are placed at strategic points whereby any shaft may be isolated from a fire in another shaft or part of the mine. Provisions are made for the immediate calling out of all superintendents and foremen at the first indication of a mine fire. Mine maps, with location of fans, normal direction of air flow, etc., are kept posted to date on file at each mine. This enables those in charge to arrive at an immediate decision in regard to such ventilation changes as closing fire doors, etc., as may be necessary.

A mine-rescue station is equipped with 65 sets of 2-hr. Paul oxygen breathing apparatus. There is a man in constant attendance with an auto truck, by which any mine can be serviced with breathing apparatus within 10 min. of discovery of a fire. Each assistant foreman and shift boss who is physically fit must take the United States Bureau of Mines training in use and care of mine-rescue breathing apparatus. All fire watchmen must also have this training and a minimum crew of at least 10 men of different occupations, such as motormen, pipemen, miners, etc., who have had the mine-rescue training, is required at each mine subject to call for fire duty.

The first consideration in case of underground fire is to get the men out of danger. Nearly all fires of recent origin, owing to the somewhat elaborate system of fire watching, have been extinguished before smoke reached the working places, but there is no hesitation in removing the men from the mine in any case.

One of the assistant general superintendents is designated as Fire Superintendent. When any fire gets away to a good start this particular superintendent takes charge, and, in consultation with the mine foreman, designates the changes in ventilation, location at which to erect stoppings, etc., and the fire area is isolated as quickly as possible. Much of this work is done under oxygen breathing apparatus. After isolation of the fire area has been accomplished the procedure varies according to circumstances. In some cases direct approach is used, in others drill holes are used to tap the fire area and flush it with water or fill it completely with mill tailings. A detailed description of these methods is given by H. J. Rahilly.[2]

SEMIRELATED DEPARTMENTS

There are three departments, in addition to those named above, that have an influence on the health and safety work. The Labor Commissioner is in charge of all contractual arrangements with the trade unions and miner's unions. He adjusts complaints in regard to contract wages, grievances in regard to alleged wrongful discharge, complaints against hospital treatment, ambulance and first-aid service. The idea of this department is to provide an open door for all to reach the management with any matter pertaining to employment.

INSURANCE

In July 1936 the company adopted a voluntary group insurance plan in which the employee pays 60¢ monthly per $1000 of insurance. This amount is deducted through the pay roll and the remainer of the cost is borne by the company. Under this plan employees are insured without physical examination and claims are paid without consideration of the cause of death. The minimum amount carried is $1250 and the maximum is $10,000, based upon the normal annual earnings of the insured employee. The amounts carried are designated by the insurance company and cannot be changed; thus, an employee normally earning less than $2000 annually carries the minimum of $1250, while one earning $2000 but less than $3000 carries $2000 insurance. When this insurance plan was offered 92 per cent of the employees accepted it at once. Arrangements are made whereby, during temporary unemployment due to illness, injury or suspension of work, employees may keep up their insurance for a period of six months by paying the premiums monthly in cash, and if they leave the employ of the company may transfer their policies to straight life policies without examination but at the regular rate designated for that age.

The group insurance plan became effective July 1, 1936 and for the Western Division up to and including the last day of November 1937, seventeen months, a total of 251 death claims amounting to $356,500.00 had been paid.

---

[2] H. J. Rahilly: Mine Fires and Hydraulic Filling. *Trans.* A.I.M.E. (1922) **68,** 61.

ENGINEERING RESEARCH DEPARTMENT

The Engineering Research Department, which originally began as the Rock Drill Service and Salvage Department, is charged with the duty of taking care of salvage and savings, care and testing of rock drills, experimentation with explosives and methods of blasting, standardization of equipment, distribution of mine timber, experimental work with rock-handling equipment, loaders, slushers, etc., testing new equipment offered by manufacturers, etc. This department encourages the offer of new ideas by employees, and when any employee suggests an improvement in any device or method, his suggestion is referred to and tested by the Engineering Research. The department has been particularly helpful to safety through its development of "bunch blasting," central mechanical fuse-capping plants, etc. These efforts have resulted in a remarkable reduction in misfires and explosives accidents.

SUMMARY

During the period of 25 years in which the company has engaged in organized effort for the health and safety of its mine employees at Butte, the following new equipment and methods have been adopted:

1. Mine-rescue oxygen breathing apparatus, rescue station and first aid training.
2. Definite and detailed reporting of all accidents.
3. Definite safety rules and inspection of work for their enforcement.
4. Regular meetings of superintendents, foremen and bosses for the discussion of safety and health.
5. General adoption of mechanical ventilation with booster-fan and vent-tubing distribution to working places and air-conditioning plants.
6. Adoption of wet drilling, involving piping of all working places with city water supply, which provides a means of sprinkling or wetting down travelways and working places.
7. Adoption of detachable bits, which eliminate daily handling of tons of steel.
8. Guarding of all machinery and falling hazards.
9. Elimination of trolley wires through adoption of battery locomotives for underground tramming.
10. Investigation and experimentation that has resulted in better grade of explosives and safer methods of handling them.
11. Elimination of open-flame lamps through the adoption of miners' battery cap lamps.
12. Reduction of head injuries through the adoption of protective hats.
13. Improvement of health conditions through modernizing and servicing change houses with central heating plants.
14. Ambulance and first-aid service to injured.

15. Mechanical rock-handling machinery such as shovel loaders, scrapers, etc.

16. Improvements in hoisting equipment, cable inspections, enclosed man cages and regulations in regard to hoisting men.

17. Improvement in supervision by increased number of shift bosses, with resultant reduction in size of crews.

18. Increased efforts in education and propaganda through safety magazines, bulletin services and National Safety Council membership.

Ventilation and Air Conditioning of the Magma Mine

By C. B. Foraker,* Member A.I.M.E.

(New York Meeting, February, 1938)

The Magma mine, of the Magma Copper Co., at Superior, Pinal County, Arizona, is 68 miles east of Phoenix and 21 miles west of Miami, Arizona, on highway U. S. 180.

Temperatures and Underground Waters

Surface Temperatures.—The average surface dry-bulb temperature at Superior is 72.4°; wet-bulb, 57.4°; with an indicated relative humidity of 38 per cent. The average yearly rainfall from 1921 to 1936, inclusive, was 18.74 in. The maximum condition of the atmosphere is in June, July and August, and the minimum is in January and February. The surface temperatures, of course, influence the temperatures of the downcast shafts; however, the variation between winter and summer causes a difference of only 2° to 3° in the stopes and working places that are not close to the intake shafts. Average atmospheric conditions are shown in Fig. 1.

Rock Temperatures.—Rock temperatures on the deeper levels of the mine are as follows:

Level	Degrees	Level	Degrees
2000	109	3200	127
2250	112.5	3400	130
2550	116	3600	134
2800	120	3800	137
3000	124	4000	140

There appears to be no difference in rock temperature for any particular horizon of the various formations or in the vein. Water encountered is always the same temperature as the rock from which it flows, and for this reason it is thought that the water is heated by the rock; not the rock by the water. The country rock apparently has been heated by the various intrusions, perhaps the vein itself, and has not yet had time to cool off. The geothermal gradient (Fig. 2) as indicated by the rock temperatures given above is 1½° per 100 feet.

Underground Waters.—The underground water varies from 350 to 565 gal. per min., depending on the rainfall. The limestone and other

---

Manuscript received at the office of the Institute Dec. 30, 1937. Issued as T.P. 979 in Mining Technology, September 1938.

* Ventilation and Safety Engineer, Magma Copper Co., Superior, Arizona.

sedimentaries act as a reservoir, and as the rain water from the surface percolates slowly through them it is eventually found underground on the contacts of the various formations, principally on the contact between the diabase and the sediments. For this reason, most of the water is found in the eastern part of the mine, where the sedimentaries have been cut by the east drifts. The vein, however, for the most part, is damp throughout.

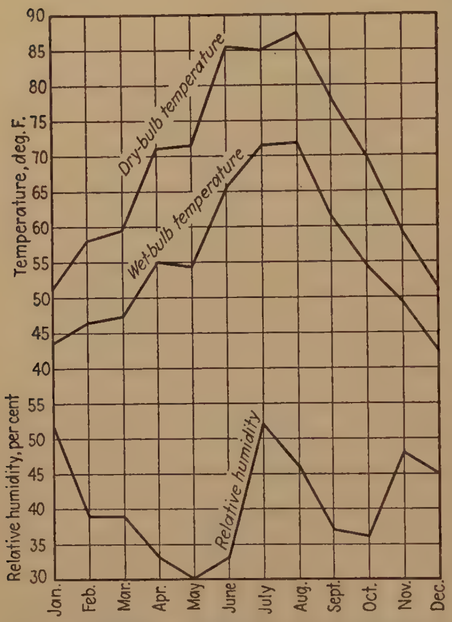


Fig. 1.—Average surface atmospheric conditions, Superior, Arizona.

For this reason the first drift driven in opening up a new level is the footwall drift, which is driven about 60 ft. from the footwall of the vein and parallel to it. The dip of the vein has been so consistent in recent years that it is possible to project it to the levels below and to drive the footwall drift in the proper position. It has now become the practice to put in an occasional diamond-drill hole from the footwall drift to the vein to check the position of the footwall drift for its relation to the vein.

If the first drift on a new level were to be driven in the vein, the dampness of the vein, with its high rock temperatures, would make such high humidity of the air that it would be impossible to work.

After the footwall drift is driven and through ventilation is established crosscuts are made every 150 ft., and the extraction drift is driven from these crosscuts.

Typical Development

An example of the above procedure is shown in the following development. When it became necessary to develop the mine below the 2800-ft. level, Nos. 2, 3, and 5 shafts were sunk to the 3200-ft. level. and No. 14 and No. 31 winzes were sunk to the same depth at approximately one-third the distance between the shafts. Nos. 2 and 3 shafts were connected on the 3200-ft. level and the footwall drift was driven from this crosscut and also from No. 5 shaft. The drift from No. 3 shaft connected with 14 winze, and the drift from No. 5 connected with 31 winze. When these connections were made, ventilation was established down No. 5 shaft and up 31 winze, and also down No. 3 shaft and up 14 winze. It was then possible to establish new sources of fresh air on the 3200-ft. level and the footwall drift was driven from each side until a connection was made.

Crosscuts were then put into the vein every 150 ft. and the extraction drift was driven in the vein. The extraction drift served to drain the vein from the 3200 to the 2800-ft. level before the 3000-ft. level was started. This made it possible to drive on the vein on the 3000-ft. level without first putting in the footwall drift.

The next step was to drive raises on 105-ft. centers from the 3200 to the 3000-ft. level, and from the 3000 to the 2800-ft. level, and to circulate air through them. They served to further cool and drain the vein that was to be mined later.

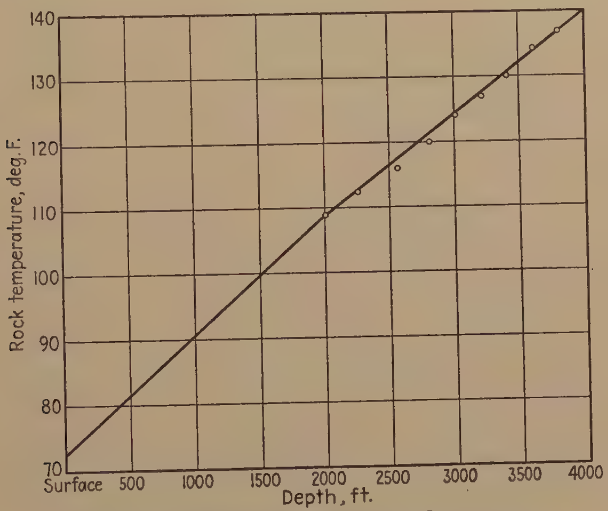


Fig. 2.—Geothermal gradient, Magma mine.

It was necessary to drive the 3200-ft. level almost four years before actual stoping operations could be started. Temperatures at the faces of the drifts on that level, while they were being driven, averaged as follows: 94° wet bulb, 105° dry bulb, and 72 per cent relative humidity. After this block had been drained and ventilated for four years, the stope temperatures averaged 82° wet bulb, 88° dry bulb, and 77 per cent relative humidity. Satisfactory working conditions were thus obtained by ventilation alone; however, the capital necessary to develop the levels was tied up four years before any return could be realized from it.

It might be noted here that it is impossible to work any place underground unless ventilation is available, either by a through ventilation current or by auxiliary ventilation with booster fans and flexible ventilation tubing.

At one point in the development of the 3200-ft. level, while ventilation was being supplied to the working face of the footwall drift by a fan on the 2800-ft. level at No. 5 shaft, the air was taken down the shaft in 16-in. metal tubing and then on to the face by 16-in. flexible ventilation tubing. The following temperatures were recorded:

Fan intake on 2800-ft. level.	70 wet bulb	74 dry bulb	82 per cent relative humidity
Fan discharge at face......	85 wet bulb	108 dry bulb	39 per cent relative humidity
Rock temperature 127°			

The working face was 2800 ft. from the fan intake on the 2800-ft. level.

VENTILATION

Down to the 2000-ft. level, all drifts and crosscuts were small, usually about 5 by 7 ft. As the necessity for better ventilation became apparent, all drifts below the 2000-ft. level have been driven 8 by 8 ft. This helps to reduce the mine resistance and permits a greater volume of air to be circulated.

In general, the ventilation system consists of taking the air to the lower levels by using booster fans near the bottoms of the shafts and then permitting the air to ascend through the working places by means of the stope raises to the exhaust shafts.

Main Fans.—Shafts 2, 3, 5, 6 and 7 are downcast and Nos. 4 and 8 are exhaust shafts. No. 4 shaft is exhausting 150,000 cu. ft. per min. by means of a fan on the 500-ft. level.

No. 8 shaft is exhausting 90,000 cu. ft. per min. from the pressure of the underground booster fans. When the shaft has been completed to the 4000-ft. level, a fan similar to the one on No. 4 shaft will be installed at the collar and it will also exhaust 150,000 cu. ft. per minute.

The No. 4 shaft exhaust fan is an 8 by 4 Jeffrey, of the forward-curved blade type with double inlets. It is operated at 250 r.p.m. by Texrope drive from a 150-hp., 2200-volt, 25-cycle, three-phase, 500-r.p.m. synchronous motor. It is passing 150,000 cu. ft. per min. at 4-in. water-gauge pressure, which is its rated capacity.

The fan to be installed at the collar of No. 8 shaft will be an exact duplicate of the No. 4 shaft fan, except that the housing will be of heavier construction. The latter is being used to allow operation at a higher water-gauge pressure if that becomes necessary some time in the future.

The No. 8 shaft will serve as an exhaust for the western part of the mine, and No. 4 will serve the eastern part, making the total circulation for the mine 300,000 cu. ft. per minute.

Booster Fans.—The location of the underground booster fans is shown on the isometric sketch of the ventilation system (Fig. 3). The principal underground booster fans for the western part of the mine are on the 3200, 3400 and 3600-ft. levels near No. 5 shaft, and draw air from both

No. 5 and No. 7 shafts. In the eastern part of the mine, booster fans are on the 2000 and 2550-ft. levels and draw air from No. 6 shaft. The 3000 east is ventilated by a fan drawing air from No. 2 shaft, and a fan on 2800 draws air from No. 3 shaft and discharges it into the stoping zone above that level. On the 3200-ft. level, a fan near No. 3 shaft draws air from both No. 2 and No. 3 shafts for the stopes above 3200.

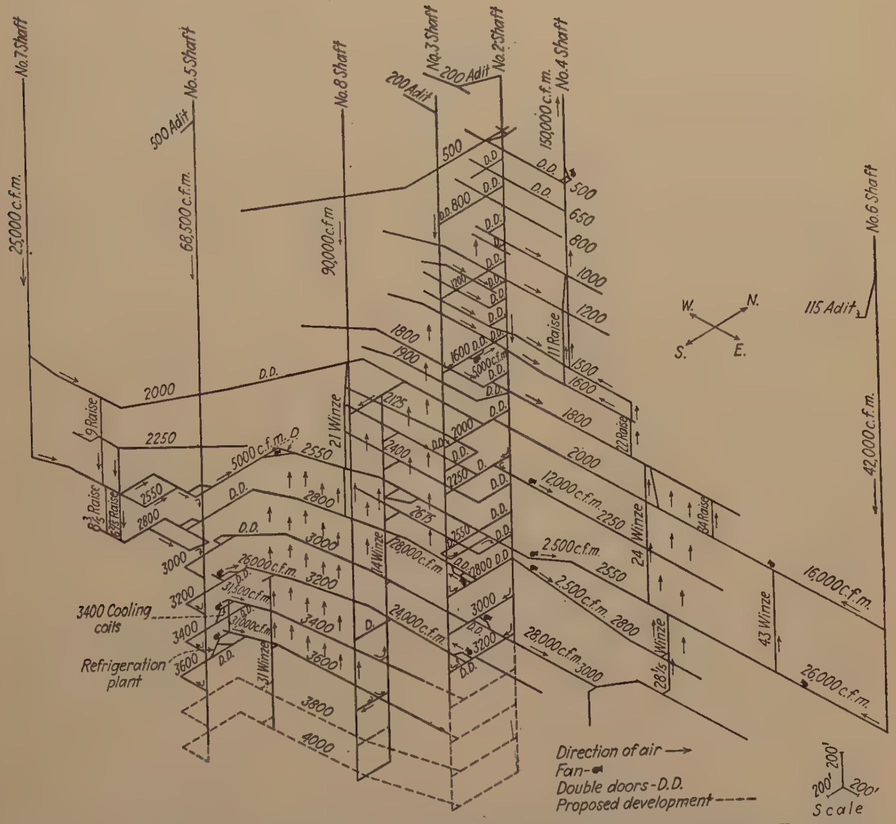


FIG. 3.—ISOMETRIC SKETCH, MAGMA MINE, SHOWING VENTILATION SYSTEM, DECEMBER 1937.

The Magma Copper Co. has standardized on the No. 8 American high-speed fans for booster service. It has been found that they are adequate, are easily handled in small shafts, and require very little rock excavation when installed. They are of the double-width, double-inlet type, with backward-curved blades giving a nonoverloading characteristic. Power is supplied by a 40-hp., 2200-volt, 25-cycle, three-phase induction motor at 720 r.p.m. They are designed to handle 40,000 cu. ft. per min. at 4-in. water-gauge pressure.

Auxiliary Ventilation.—It is necessary to ventilate every working place, either by through ventilation or by auxiliary ventilation. For the latter, two types of small fans are used. The No. 3 Sirocco fan of the multiblade type is used when it is not necessary to conduct the air a great distance. Ordinarily it is used for ventilating raises that are being driven, one fan being capable of ventilating two raises by means of flexible ventilation tubing of 12-in. diameter. Where it is necessary to conduct the air from 1000 to 3000 ft., as in driving long drifts, the No. 4½ American high-speed fan is used. This delivers approximately 5000 cu. ft. per min. against a 5-in. water-gauge pressure. For long installations, 16-in. galvanized-iron conduit is used. This tubing is made up at the mine in 16-ft. lengths and the sections are fitted together with beaded collars over cheesecloth that has been painted with tar, thus making an air-tight joint. For shorter installations, rubberized canvas tubing is used, either of 16 or 24-in. diameter, depending on the volume of air necessary at the working place.

Costs.—Table 1 shows ventilation costs at the Magma mine for the period 1931 to 1936, inclusive.

Table 1.—*Ventilation Costs, Magma Mine*

Year	Labor	Power	Supplies	Total	Unit Costs	
					Cents per Ton Ore	Cents per Lb. Copper
1931	$ 9,381.55	$ 13,413.15	$ 3,782.42	$ 26,577.12	0.114625	0.000888
1932	7,606.15	17,866.73	3,418.20	28,891.08	0.193887	0.001308
1933	7,883.05	18,613.88	4,977.29	31,474.22	0.216429	0.001413
1934	10,313.71	17,052.25	10,792.73	38,158.69	0.144489	0.001164
1935	8,979.06	17,743.03	12,591.22	39,313.31	0.151465	0.001230
1936	10,306.58	19,346.62	11,375.81	41,029.01	0.149706	0.001257
	54,470.10	104,035.66	46,937.67	205,443.43		
Mean per cent....	26.51	50.64	22.85	100		

Results Obtained by Ventilation.—The most favorable surface temperatures are obtained during the month of February, which shows the following average conditions: wet-bulb, 47½° and dry-bulb, 58°, with an average relative humidity of 39 per cent. With these surface temperatures and while circulating 1200 cu. ft. per min. per man underground, the average temperature of 51 working places in February 1937 was wet-bulb, 81.5° and dry-bulb, 88¼°, with relative humidity 75 per cent. The poorest surface conditions occur in the month of July, which shows the following average temperatures for the month: wet-bulb, 71°; dry-bulb, 85½°; relative humidity, 52 per cent. While circulating the same amount of air as in the month of February 1937 the following temperatures were recorded underground in July 1937: average of 43 working places, wet-bulb, 89°; dry-bulb, 93½°; relative humidity, 83.5 per cent.

Other Precautions

It has been the experience at Magma that the efficiency of the workmen falls off rapidly when the dry-bulb temperature is above 85° and the relative humidity is above 85 per cent. Wet drilling and the sprinkling of muck piles to allay the dust is, of course, necessary, even though it tends to raise the relative humidity. Every effort is made to keep drifts and tunnels as dry as possible by confining water to ditches and keeping the water from coming in contact with the atmosphere.

In the past, occasional cases of heat cramps occurred, caused by loss of salt from the body of the workman through excessive perspiration; but none were serious. Early in 1936, 15-grain salt tablets were made available to all workmen. They are taken orally, usually with a drink of water, and they maintain the proper saline balance in the body. Since this practice was instituted, no workman who has used the salt tablets has had an attack of heat cramps.

None of the fans have reversing facilities, it being thought best to maintain all air currents as nearly normal as possible in case of fire. For giving underground fire alarms, ethyl mercaptan is used. This chemical is injected into the compressed-air lines, from which it passes into the ventilation currents and reaches all working places. It is extremely volatile and vaporizes immediately when liberated, and has the most unpleasant odor imaginable. By this method, a fire alarm may be given to everyone underground in less than 20 minutes.

Air Conditioning

After the 3200-ft. level had been developed, it became evident that artificial cooling and air conditioning would be necessary to assist the ventilation. Early in 1936 it was decided to install a refrigeration plant on the 3600-ft. level to air-condition the 3600 and 3400-ft. levels. Plans of these levels are shown in Figs. 4 and 5.

For this refrigeration plant, two Carrier centrifugal compressor units, each of 140 tons refrigeration capacity, were selected. The refrigerant used is Carrene No. 2, monofluorotrichloromethane ($CFCl_3$), which is a colorless, odorless liquid at ordinary temperatures. It is nontoxic, noncombustible, noninflammable, and has a boiling point of 74°. Each unit is powered with a 200-hp., 2200-volt, three-phase, 25-cycle, induction motor operating at 1440 r.p.m., which is stepped up by speed-increasing gears to drive the compressor at 6750 r.p.m. (Fig. 6).

When each unit is furnished with 200 gal. per min. of 90° condenser water, it will cool 350 gal. of chilled water to 60°. This chilled water is circulated in closed circuit through Aerofin cooling coils (Fig. 7). The chilled water from one unit is pumped to a bank of cooling coils on the 3400-ft. level, and from the other unit to cooling coils on the 3600-ft. level.

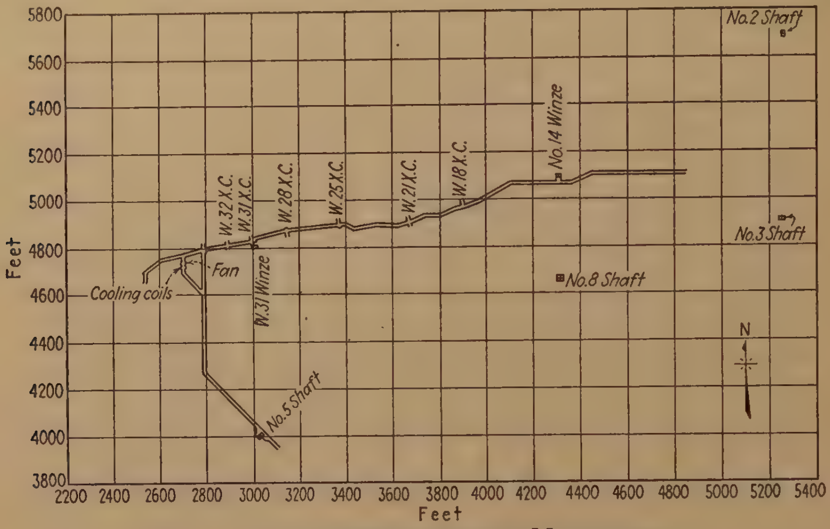


FIG. 4.—PLAN OF 3400-FOOT LEVEL, MAGMA MINE.

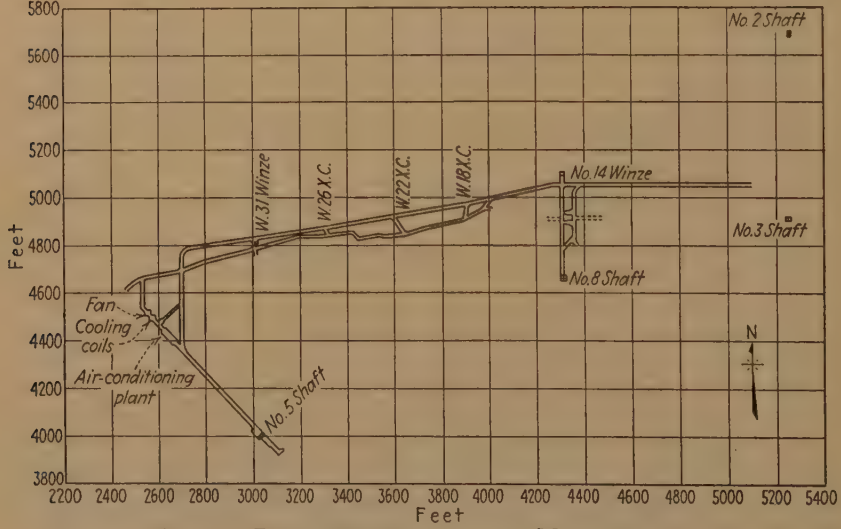
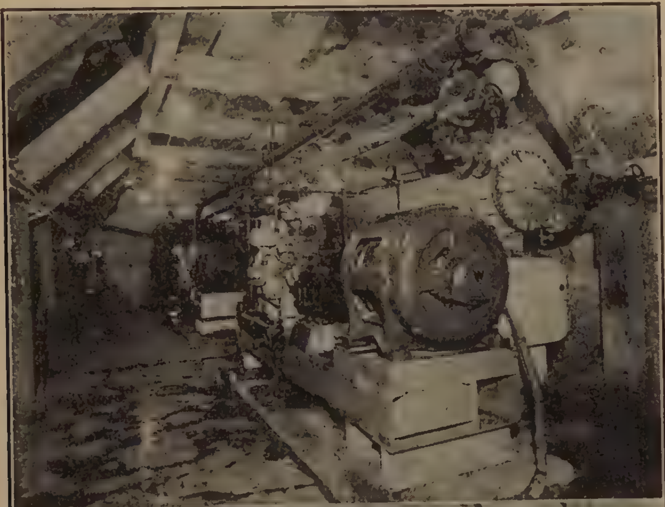


FIG. 5.—PLAN OF 3600-FOOT LEVEL, MAGMA MINE.

Fans of 30,000 cu. ft. per min. capacity draw air through the cooling coils and the coils are designed to cool this volume of air 12° on the wet-bulb temperature.

Cycle of Operation of Refrigeration Units.—The cycle of operation may be traced by starting at any point in the system. Starting with the chilled water at the refrigeration unit, the cycle is as follows: 350 gal. per min. of water is passed through tubes in the cooler or evaporator, where it is cooled to a temperature of 60° or lower. This water is pumped in closed circuit to the cooling coils, where it makes four passes


6

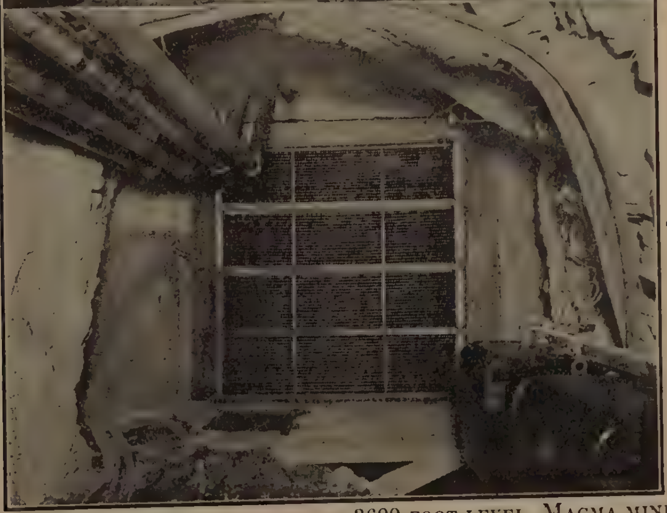

7

FIG. 6.—REFRIGERATION PLANT ON 3600-FOOT LEVEL, MAGMA MINE. Cooling coils for this level may be seen in extreme background of picture.

FIG. 7.—COOLING COILS ON 3600-FOOT LEVEL.

through copper tubing with extended surfaces or fins, and heat is transferred from the air to the water, and the water is returned to the evaporator for re-cooling. (The air to be cooled is drawn through the cooling coils by means of a fan.) The refrigerant in the base of the evaporator is in liquid form and the evaporator is operated under a vacuum of 11½ to 14 lb. per sq. in., which is maintained by the centrifugal compressor. The refrigerant is pumped up and allowed to spray over the tubes in the evaporator or cooler, and at this low pressure (i.e., vacuum) it boils or evaporates rapidly, absorbing its latent heat of evaporation from the water to be cooled. The refrigerant, now in vapor form, passes through the centrifugal compressor, where it is compressed to 20 lb. per sq. in.; then it passes into the condenser and circulates about the tubes through which the condenser cooling water flows. In the condenser, the vapor gives up to the condenser cooling water its heat of compression and latent heat of liquefaction. The condenser water, containing the heat, is then removed. The vapor in the condenser again becomes a liquid and is returned to the evaporator, thus completing the cycle.

Condenser Water.—The primary problem presented when air conditioning became necessary was to obtain a large enough supply of water that was cool enough for condenser purposes. Many unsuccessful attempts have been made to develop water, both on the surface and underground, but the only water available was the underground water, which averages about 400 gal. per min. yearly. Therefore it became necessary to use the entire underground supply for condenser water.

The next serious problem was the temperature of the underground water. At the various points where it issues from the rock in the eastern part of the mine from the 2000 to the 3000-ft. levels, the temperature is from 109° to 130°. It has therefore been necessary to sacrifice a portion of the ventilation air in the eastern part of the mine to cool the water to 90°. This is done by spraying the water down several winzes and blowing air up the winzes against the spray.

The third serious water problem was the distance from its source to the refrigeration plant. The water is brought in ditches from the eastern part of the mine to No. 2 and No. 3 shafts and is then dropped to the 3200-ft. level, where it is collected in a sump. From this sump it is siphoned to the refrigeration plant on the 3600-ft. level.

From past experience, it has been found that copper condenser tubes stand up better under mine water than those of any other material, therefore copper tubes were used. After four months, the tubes are apparently standing up and a proper heat transfer is being obtained. It is not yet known how often it will be necessary to clean the tubes.

Pipe Installation.—To pipe the condenser water from the sump on the 3200-ft. level to the refrigeration station and then to the main underground pump station at the No. 3 shaft on the 2550-ft. level, it was necessary to install 147 ft. of 4-in. pipe, 7172 ft. of 6-in. pipe and 610 ft. of 8-in. pipe, and 2000 ft. of the 6-in. pipe had to be insulated with 1-in. asbestos covering because it is the return condenser water line, which passes through the area to be cooled.

TABLE 2.—*Final Cost of Air-conditioning Plant, Magma Mine, October* 31, 1937

	Labor	Supplies	Total
Drifts and crosscuts—labor and supplies not charged to capital account:			
Drifts and crosscuts:			
3600-ft. level, No. 5 shaft ventilation X.C.N.	$ 2,291.07	$ 603.84	$ 2,894.91
3600-ft. level, No. 5 shaft ventilation X.C.S.	303.48	122.04	425.52
3400-ft. level, No. 5 shaft ventilation X.C.N.	2,091.25	498.40	2,589.65
Total drifts and crosscuts	4,685.80	1,224.28	5,910.08
Stations:			
3600-ft. level, No. 5 shaft fan station	270.04	82.31	352.35
3400-ft. level, No. 5 shaft fan station	88.99	12.48	101.47
Total stations	359.03	94.79	453.82
Fans (2) complete with motors and starters		3,558.00	3,558.00[a]
Total charged to development and ventilation in connection with cooling system not capitalized	5,044.83	4,877.07	9,921.90
Capital charges:			
Excavation costs:			
Cooling-plant station, cutting	1,346.12	364.12	1,710.24
Cooling-plant station, guniting	46.65	29.62	76.27
Coil-plant station, 3600-ft. level, cutting	342.43	40.55	382.98
Coil-plant station, 3600-ft. level, guniting	17.18	14.92	32.10
Coil-plant station, 3400-ft. level, cutting	252.85	54.03	306.88
Cutting 2550-ft. level sump	3,026.70	800.55	3,827.25
Cutting 3600-ft. level sump	1,257.46	278.46	1,535.92
Cutting 3600-ft. level pump station	228.83	60.84	289.67
Total excavation cost	6,518.22	1,643.09	8,161.31
Cost of equipment and installation:			
Refrigeration unit		37,367.00	37,367.00
Two Byron-Jackson pumps		1,588.10	1,588.10
Installation, refrigeration units	2,253.98	862.17	3,116.15
Total refrigeration-unit cost and installation	2,253.98	39,817.27	42,071.25
Condenser-water lines and pump installation	8,647.90	15,703.37	24,351.27
Chilled-water lines and pump installation	1,505.20	2,295.69	3,800.89
Total condenser and chilled-water lines and pumps	10,153.10	17,999.06	28,152.16
General	1,287.89		1,287.89
Power lines	1,017.18	5,801.48	6,818.66
Total cost of cooling system, 1937	21,230.37	65,260.90	86,491.27
Labor on condenser line, 1936	46.73		46.73
Final cost of cooling system, capital account	21,277.10	65,260.90	86,538.00
Final cost of cooling system, drifts, crosscuts, stations and fans not charged to capital account	26,321.93	70,137.97	96,459.90

---

[a] These two fans and other charges, while essentially a part of the system are not charged to it, as they are necessary and primarily a part of general ventilation, and by being such are charged against ventilation.

The return condenser water, which is 113° in temperature, is now being pumped out of the mine through No. 3 shaft, which is downcast, and radiates some heat to the incoming air. When the No. 8 shaft is completed, this water will be pumped up this shaft in one lift from the 3600-ft. level to the surface. A pump station is being cut on the 3600 level near No. 8 shaft and it is expected to have the pumps and column installed early in 1938. The hot water will then assist the ventilation by radiating heat and forming a hot air column, as this shaft is to be an exhaust.

For circulating the chilled water from the refrigeration plant to the cooling coils on the 3400 and 3600-ft. levels, 1650 ft. of 4-in. pipe was installed, making the total of 9579 ft. of 4, 6 and 8-in. pipe.

Rock Excavation.—To house the refrigeration units, a station 20 by 62 by 12 ft. high was cut, requiring the excavation of 14,352 cu. ft. of rock. The fan, cooling coil, and pump stations on the 3600-ft. level required an additional 8956 cu. ft. of excavation. On the 3400-ft. level, 5907 cu. ft. of rock was excavated for a fan and cooling-coil stations. On the 3600-ft. level, adjacent to the refrigeration station, a condenser water sump 8 by 8 by 108 ft. was cut, and on the 2550-ft. level a sump 6 by 7 by 255 ft., this also being to handle condenser water.

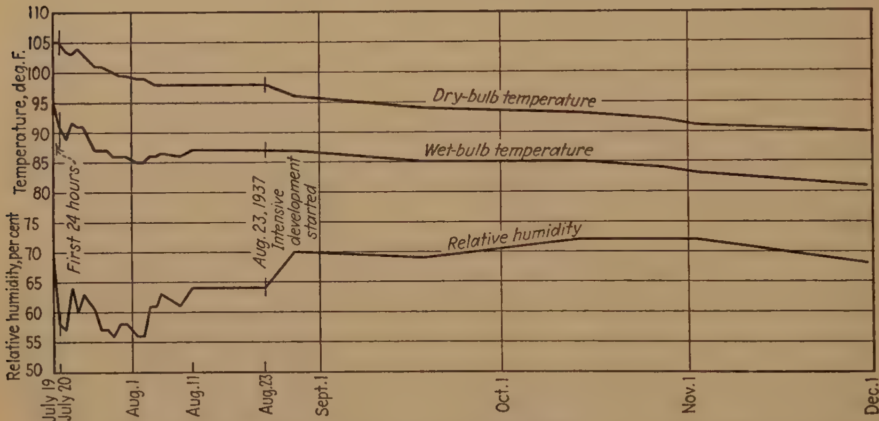


FIG. 8.—TEMPERATURE DROP AT NO. 14 WINZE ON 3600-FOOT LEVEL, 2000 FEET FROM AIR-CONDITIONING PLANT.

Size and Cost of Equipment.—All of the machinery had to be lowered through shaft compartments that are 40 by 60 in. in size. For this reason, two units of 140 tons refrigeration capacity, rather than one large unit, were selected. This also gives the plant additional flexibility, as one unit can be operated while the other is being cleaned. By changing operating valves, either unit can be connected with either set of cooling coils. Table 2 gives the cost of the air-conditioning plant.

Before and After Air Conditioning.—Table 3 shows temperatures at key points on the 3600-ft. level before air conditioning, and at one, two, three, and four months after the air-conditioning plant was started.

TABLE 3.—*Temperatures Before and With Air-conditioning*

Temperature Recorded at	July 18			July 19			July 26			Aug. 19			Sept. 19			Oct. 19			Nov. 19		
	W B[a]	D B[a]	R H[a]	W B	D B	R H	W B	D B	R H	W B	D B	R H	W B	D B	R H	W B	D B	R H	W B	D B	R H
Surface at 8:00 a.m.	68	80	54	68	80	54	66	85	36	68	79	57	59	84	19	58	69	51	40	55	19
Entering cooling coils	84	90	78	85	91	78	83	89	78	85	92	75	82	89		82	90	71	78	85	73
Leaving cooling coils	86	90	85	72	73	95	72	73	95	77	78	96	71	72	95	71	73	91	65	66	95
Fan discharge	86	94	72	75	80	79	74	78	83	78	81	88	72	76	82	72	75	86	66	70	81
3600-ft. level:																					
West drift at main crosscut	88	95	76	80	88	70	77	83	80	80	83	88	74	78	83	73	77	83	67	73	73
West drift at 31 X-cut	95	100	83	86	98	61	81	92	62	83	89	78	80	87	74	78	85	73	75	82	72
West footwall drift at 31 X-cut	90	99	70	85	96	63	81	90	68	83	89	78	78	85	73	77	84	73	75	82	72
West footwall drift at 26 X-cut	92	100	73	88	98	67	83	93	66	85	92	75	80	87	74	80	87	74	77	84	73
West footwall drift at 22 X-cut	93	102	71	89	100	65	85	96	63	86	94	72	83	90	74	82	90	71	78	86	70
West footwall drift at 18 X-cut	94	104	69	92	103	66	87	99	62	87	96	69	83	92	69	83	91	72	80	89	68
West footwall drift at 14 winze station	95	105	66	93	103	71	88	101	60	88	98	67	85	94	69	86	93	75	81	90	68

Temperatures for:
July 18 were taken before refrigeration unit was started.
July 19 were taken 2 hr. after refrigeration unit was started.
July 26 were taken after refrigeration unit had been running one week.
Aug. 19 were taken after refrigeration unit had been running one month.
Sept. 19 were taken after refrigeration unit had been running two months.
Oct. 19 were taken after refrigeration unit had been running three months.
Nov. 19 were taken after refrigeration unit had been running four months.

[a] Wet-bulb and dry-bulb, degrees; relative humidity, per cent.

The proper amount of condenser water (400 gal. per min.) was not supplied to the plant until the middle of September, when the siphon was properly regulated to handle this amount. This water at first was from 92° to 94° in temperature instead of the 90° required for best operation. However, the plant functioned very well and produced from 280 to 290 tons of refrigeration. In November, the condenser water was lowered to 90° and with this water the plant has produced as high as 306 tons of refrigeration, which is 26 tons above its rated capacity. The air entering the cooling coils is 7° cooler, wet bulb, in November than in July, because the surface atmosphere is cooler.

The 3600-ft. west footwall drift at 14 winze station is approximately 2000 ft. from the cooling plant and the temperatures have been lowered from 95° wet-bulb, 106° dry-bulb, 66 per cent relative humidity, to 81° wet-bulb, 90° dry-bulb, 68 per cent relative humidity, in four months time (Fig. 8).

A typical refrigeration log of the plant is shown as Table 4. An operator is on duty at all times and the plant is operated 24 hr. a day and 7 days a week.

TABLE 4.—*Refrigeration Log, Magma Copper Company*
UNIT ON 3600-FT. LEVEL

Attendant	Time	Cooler				Condenser			Compressor Bearings		Oil Pressure		Temp. Entering Coils		Temp. Leaving Coils		Remarks KWH 4311
		Vacuum	Liquor Temp.	Water Temp.		Pressure	Water Temp.		Suction End Temp.	Disc. End Temp.	Oil Res.	Seal	W. B.	D. B.	W. B.	D. B.	
				In	Out		In	Out									
Cluff	8:00	11.5	46	60.5	52	20	90	112	139	152	18.5	14.5	76.5	84.5	65	66	
	10:00	11.5	47	61.5	53	20	90	112	138	153	18.5	14.5	76.5	84.5	66.5	67	
	12:00	11.5	47	61.5	53	20	90.5	112	137	154	18.5	14.5	76.5	84.5	67	67.5	
	2:00	11.5	47	61.5	53	19.5	90	111	137	154	18.5	14.5	77	85	66.5	67.5	
Noetzel	4:00	11.5	47	61	53	20	90	112	137	154	18.5	14.5	76.5	84.5	66	67.5	
	6:00	11.5	47	61	53	20	90.5	112	138	155	18.5	14.5	77	84	66.5	67.5	
	8:00	11.5	47	61	53	20	90	112	138	154	18.5	14.5	77	84.5	66.5	67.5	
	10:00	11.5	47	61	53	19.5	90	111.5	138	155	18.5	14.5	76.5	84.5	66.5	68	
Brown	12:00	11.5	47	61	53	20	90	112	138	155	18.5	14.5	76.5	84.5	66.5	67.5	
	2:00	11.5	46	60	52	20	90	112	138	156	18.5	14.5	76.5	84.5	66.5	67	
	4:00	11.5	46	60	52	20	90	112	138	156	18.5	14.5	77	84.5	66.5	67.5	
	6:00	11.5	46	60	52	20	90	112	138	156	18.5	14.5	77	84.5	66.5	67	

Power Consumption and Operating Cost.—The following data on power consumption and operating costs are for the month of November 1937, which was a normal month. The compressors consumed 221,760 kw-hr.; pumps returning condenser water to the 2550-ft. level 137,555 kw-hr. and the fans, 34,995 kw-hr.; a total of 494,310 kw-hr. Power consumption per ton of ore mined was 15.93 kilowatt-hours.

The cost of operating the plant, exclusive of power, was as follows: for labor, $685.74; supplies, $17.33; cost per ton ore, $0.0226.

Results Obtained by Air Conditioning.—The temperatures in Table 5 are averages for all working places on the levels indicated, and show the improvement that has been made in a four months period of air conditioning. Part of the improvement, of course, is due to the cooler air from the surface and to ventilation alone; but the greater improvement is due to air conditioning. This is proved by the fact that the air that ascends from the 3400-ft. level to the 3200-ft. level is 4° cooler (wet bulb) than the air that comes down to the level through No. 5 shaft and is delivered to the level by a fan near the shaft on that level.

TABLE 5.—*Decrease in Temperature Caused Largely by Air Conditioning*

Level	July 15, 1937			Nov. 15, 1937		
	Wet-bulb, Deg.	Dry-bulb, Deg.	Relative Humidity, Per Cent	Wet-bulb, Deg.	Dry-bulb, Deg.	Relative Humidity, Per Cent
3600	93	101.5	74	78.5	88.5	64
3400	90	103	58	72	80	68
3200	83.5	88.5	81	79.5	87	72
3000	89.5	91.5	92	85	90	81

In the four-month period immediately following the air conditioning, the following development and stoping has been done on the 3400 and 3600-ft. levels.

8 by 8 drifts and crosscuts driven, lin. ft.	2,590
5 by 10 raises driven, lin. ft.	973
Ore stoped (exclusive of development), tons	5,737
Stations cut (including pump stations, raise stations, sumps, etc.), cu. ft.	25,665

This means that the air cooling has not only cooled the additional rock surface exposed by the work done but has also effected a continual dropping in temperature of all working places on these two levels, and still permits the air to ascend to the working places above (3200 and 3000-ft. levels) at a lower temperature than air can be brought directly to the levels.

The psychological effect and the improved morale of the workmen are factors that cannot be accurately gauged, but are present nevertheless.

The improvement in working conditions has been even greater than was anticipated and the results are extremely gratifying.

Ventilation and Dust Prevention in the Butte Mines

By A. S. Richardson,* Member A.I.M.E.

(New York Meeting, February, 1938)

Ventilation of the Butte mines has long been a rather difficult problem because of the natural high temperature of the rock. With increase in mining depth, higher rock temperatures have been encountered, and this has caused a constantly growing demand for more effective ventilation measures. During recent years the hazards of dust in mine air have been more clearly recognized, and have necessitated the adoption of dust-prevention methods.

To meet these ventilation problems in the mines of the Anaconda Copper Mining Co., a mines ventilation department was organized in 1918, to act under the direction of the general superintendent of mines. At the present time, under normal operating conditions, the force employed includes the ventilation engineer, two chief assistant ventilation engineers and eight assistant ventilation engineers. The activities of the department are, briefly: (1) planning and supervising operation of the general ventilation system; (2) planning and supervising the operation of dust-prevention measures, together with dust sampling and related microscopic work for the quantitation of the dust samples and determination of particle-size distribution; (3) the design and development of air-conditioning methods to meet present exceptionally severe temperature conditions and in anticipation of future requirements.

To a large extent the technical features of the ventilation methods have been described in earlier papers by the author. This paper is written mainly to summarize past accomplishments bearing most closely on health and safety, with a somewhat more detailed account of the dust-prevention measures that have been developed comparatively recently.

FACILITATING FLOW OF AIR

As a basic requirement for the safety of the men, all ventilation plans are subject to a ruling that the main operating shafts must be maintained as inlet air courses. Under this condition, a safe, fresh-air escapeway is assured if a fire occurs within the mine, and the shaft is available as a base

---

Manuscript received at the office of the Institute Jan. 22, 1938. Issued as T.P. 969 in Mining Technology, September 1938.

* Ventilation Engineer, Anaconda Copper Mining Co., Butte, Mont.

for fire-fighting operations. Control of the direction of air flow is provided by the use of reversible fans located at the surface of the outlet air shafts, and regulation of the volume of air may be obtained by control dampers in the fan ducts.

Although mechanical ventilation of the mines has been the common practice for many years, the continual increase in mining depth rendered the equipment in use in 1918 inadequate to maintain a satisfactory standard of ventilation conditions. To a large extent this was due to increases in both temperature of the rock and the resistance to air flow that is offered by the mine workings. Improvement of conditions depended upon an increased flow of air through the mines, but since power requirements vary approximately as the cube of the volume of air, it was necessary to devise better means for economically obtaining such an increase.

To meet this problem a study of the coefficients of frictional resistance to the flow of air that is offered by different types of mine workings was made in 1917 and 1918, and is believed to have been the first ever made in metal mines. From these data calculations were made as to the benefit that might be obtained by smooth-surfacing the outlet air shafts in order to reduce frictional resistance. Estimates of the flow that might then be obtained with new and more efficient fans were made, and a ventilation improvement program was commenced.

Under this program a reduction in mine resistance by smooth-surfacing work in 20 air shafts was carried out, and at the same time 23 new, reversible, and more efficient main surface fans were installed.

On completion of this work the total volume of air circulated through the mines by the main surface fans was increased from approximately one and one-half million cubic feet per minute to slightly over three million cubic feet per minute. Assuming no change in efficiency, the power required to effect such an increase in flow would have been approximately 800 per cent. The actual increase was approximately 70 per cent.

As a further part of the same program, a number of booster fans were installed underground to assist in boosting the air to the sections of the mine where it was most needed, and where resistance to flow was relatively highest. At the same time plans were made for the ventilation of dead ends and other places with insufficient circulation of air by means of auxiliary ventilation equipment. To accomplish this result blowers were placed at the nearest source of fresh air and lines of flexible tubing were carried from the blowers to deliver the air to the working places. For this service 300 blowers of special design were ordered, and a shop was opened to make up the flexible tubing. Owing to the destructive effect of copper water, large quantities of flexible tubing are continuously required, and the total footage sent to the mines has run as high as 20,000 linear feet per month.

Expenditures made under this program for new equipment, work on the mine shafts and connecting air courses, totaled approximately $3,000,000.

DECREASE IN TEMPERATURE AND HUMIDITY

As a result very appreciable decreases in the temperature and humidity of the air in the working places were effected in all mines. At one mine, on the basis of conditions in active stopes, crosscuts, drifts, and raises, the average mine temperature of 85.8° F. and 83.0° F. wet bulb, was decreased to 76.2° F. and 67° F. wet bulb. At different properties various degrees of improvement were noted, and in general the average mine temperatures were lowered by from 6° to 10° F., while the wet-bulb temperatures were lowered by 10° to 17° F. The total number of working places thus affected would be roughly 1000.

Since completion of this program there has, of course, been a continuous further gradual deepening of the working horizon, with an accompanying increase in rock temperatures. With this, a further increase in mine resistance has occurred, and a multiplication of ventilation difficulties caused by air leakage. To meet these conditions a constant improvement in methods and equipment has been necessary and no ventilation problem is ever permanently solved.

CONSTRUCTION OF SHAFTS

In the extension of the air shafts to greater depth, the old conventional, square-set, multiple-compartment type of shaft has been abandoned, and for it there has been substituted an octagonal opening supported on skin-to-skin timber. This affords a smooth-surfaced opening ideally adapted to service as an outlet air course. On the basis of equivalent air-carrying capacity, the cost of both excavation and timber is about 40 per cent less than that of the rectangular, smooth-surfaced, shaft. Initial cost is somewhat lower than that of a circular concrete shaft, and in the matter of repairs the timbered octagon is better adapted to ground conditions existing in the district.

In many places the vertical line of the original shaft has been broken and the octagonal extension carried at an angle to the horizontal designed to reduce to as low a figure as possible the cost of driving connecting crosscuts from the stoping areas.

FANS

In general, but few changes have been made in reversible surface fan equipment. In one place, however, where a new fan was required a high-speed propeller was substituted for the centrifugal type. By use of a propeller fan a reversal of the direction of the flow of air may be obtained by reversing the rotation of the propeller through electrical controls, and thus the cost of providing air ducts and doors for reversing air flow with centrifugal fans is eliminated. For this installation it was required to deliver 300,000 cu. ft. of air per minute against a resistance pressure of 9 in. of water. The propeller used is 6 ft. in diameter and is driven direct-connected to a 1760 r.p.m. motor. Although the cost of the propeller was higher than the centrifugal fan, the total cost of the installation was 40 per cent less. By this installation the average temperature of the working places in the mine was lowered approximately 6° F.

Increase in resistance to air flow at other mines has been largely overcome by the use of booster fans operating in series with the main surface fans to develop more pressure. The total number of such fans in service has risen from about 12 in 1922 to 42 at the present time, and the total volume of air now circulated by them is approximately 2,000,000 cu. ft. per minute. Improved types of booster fans have been obtained from the manufacturers, which have eliminated many operating difficulties formerly experienced. Most of the old type centrifugal fans, with forward-tipped blades, which were belt-driven by electric motors, have now been replaced by high-speed fans with backward-tipped blades that have flat power characteristics and are direct-connected to the motors.

For the auxiliary ventilation of dead ends a much greater number of blowers is also required. Earlier practice in the ventilation of these places was to base requirements upon temperature and humidity conditions alone, but auxiliary equipment is now installed irrespective of such conditions and solely for the purpose of preventing high concentrations of dust. A smaller blower of greater power has been developed and the total number of blowers in service was recently nearly 600.

AIR CONDITIONING

Methods of mine ventilation that depend upon the cooling power of air drawn from the surface are subject to more than ordinary limitations in the Butte district. Because of heavy, shifting ground, the mine openings that may be maintained are necessarily small, so that the volume of flow is restricted. Also, the air becomes so heated in its flow through these long, small-sized openings that its capacity for improving conditions in the working places is greatly reduced before it reaches the operating zone. A need for some method that would not be subject to these limitations and adapted to local conditions has long been apparent, and the development of such a method has been the subject of research work during the past six years. As a result of the operations of the air-conditioning plants, the average temperature of fairly large sections of a mine were reduced approximately 14° F., and in certain places temperature reductions of over 20° F. were effected.

By the ventilation methods that have been mentioned, with ground temperatures ranging up to 120° F., the temperatures of the working places in the mines as of August 1937 were kept within a range from 51° F. as a minimum to 92° F. as a maximum, with an all-mines average of 81.6° and 78.2 wet bulb.

Dust Prevention

Some recognition of the hazards of dust in mine air has, of course, always been present in the consciousness of those engaged in mining operations, and, to some extent at least, some measures for preventing dust have been used at all times. Perhaps the most important factor in preventing high dust concentrations in mine air is the provision of an adequate ventilation system to dilute the dust concentrations in the working places, and finally sweep them out of the mine. As a means of preventing the generation of dust, the most important measure taken in the Butte mines was the adoption of wet drilling methods in 1918. Other common precautions, such as the wetting down of the working places and muckpiles, are of undoubted value, but would be to a great extent ineffective to prevent high dust concentrations without control of dust generated during drilling and thorough ventilation.

In all efforts to attack dust hazards in the Butte district, much assistance has always been received from the U. S. Bureau of Mines. In 1935 an invitation was extended to the Bureau to make an examination of existing conditions and offer recommendations for improvement. A cooperative study of conditions was made and a program of research work was carried out, which covered such subjects as comparisons of results obtained by different methods of determining dustiness in mine air, the relative importance of different causes of dust, and the best methods of prevention. Publication of the results of this work is being made by the Bureau.

Details of dust-prevention measures now practiced, most of which have been in use for some time, are briefly as follows:

Around the collars of the operating shafts, or main inlet air courses, the roads and yards are oiled.

Water sprays and compressed air-water blasts are installed in the operating shafts at different elevations and used to maintain a moist condition.

Water sprays with remote control are provided at skip-loading chutes and station tenders are provided with respirators, though no high dust concentrations ordinarily occur during loading because the ore has been previously wet in the mine.

Shaft stations must be wet down and cleaned at the start of the shift, and periodically wet down and cleaned as required thereafter.

All main haulageways and inlet air courses are periodically wet down, different methods being used under different mine conditions. In some places a large sprinkling tank is used; in others, permanently fixed sprays; and on some levels a hose is carried by a man whose sole duty it is to wet down certain areas. Fixed hose are also used at other places.

Tops of all raises in active zones are provided with short sprinkling hose. Main-raise travelways are provided with sprinkling hose at all offsets, or else atomizing water sprays handling small quantities of water.

A general rule has been made requiring the wetting down of all working places at the start of the shift before any other work is done, and special sprinkling hose are provided for the purpose.

All dead-end places are equipped with blower and flexible tubing for auxiliary ventilation, and the end of the flexible tubing must not be more than 25 ft. from the face. In drifts and crosscuts the air is discharged from an oblong discharge piece, which causes it to flow in a tabular stream that does not impinge on the men's bodies but strikes the mine back slightly ahead of the breast. The air stream then follows down the face and out along the floor.

In raises special precautions have been found necessary. Chute sides next to the manway are solidly lagged or bricked. Spaced bricking next to the manway has been eliminated. An individual blower, with flexible tubing, is installed for each raise, and the tubing must be carried up to within 8 ft. of the back. Grated safety landings are used to facilitate ventilation. In three-compartment raises the compressed air-water blasts have two discharge openings.

As a result of research work, it was found that the dust disseminated into the air during drilling might be greatly reduced by increasing the quantity of water forced through the drill steel to the cutting edges of the bit. Changes in all stoper drills have been made, therefore, which will approximately double the flow of water through the steel.

Determining Characteristics of Dust

While the objective of all dust-control work is, very simply, the prevention of high concentrations of dust in the mine air, the measurement of the results obtained in this work depends upon methods of determining the characteristics of dust concentrations that are neither simple nor reliable, either quantitatively or qualitatively. Methods giving widely different results are used in different mining districts all over the world, and comparisons of results obtained by these different methods mean almost nothing.

In the earlier work done by this company in the Butte district, the instrument used was the standard impinger sponsored by the U. S. Public Health Service and operated by means of a hand pump. Later on a considerable amount of work was done with konimeters similar to those used in the mines of South Africa and Canada. More recently the instrument generally used, particularly for research work, has been the standard impinger operated by compressed air. This equipment is, however, very cumbersome for mine use, so that for routine inspections a change is being gradually made to the use of a midget impinger, which is one-tenth the size of the standard impinger. This instrument is operated by means of a hand pump, so that it is independent of compressed-air supply, and it is small enough to be conveniently carried through a mine. Samples taken simultaneously with the standard and midget impingers usually check within about 15 per cent. Of course, great care must be taken in the calibration of all apparatus used in this work.

For determining the particle-size distribution of the dust, the instrument used in taking the sample is the Owens jet dust counter with special head for the taking of multiple samples on an oblong slide.

Microscopic equipment includes a microscope with special stage for the examination of konimeter samples. No heat or acid treatment is given the slides for the removal of nonsiliceous particles, and the count is made by light-field illumination at a magnification of 200 diameters.

For the quantitation of impinger samples, which are taken in propyl alcohol, a microprojector is used. The counts are made by light-field illumination at a magnification of 1000 diameters. This method closely checks the ordinary microscope technique; is about twice as fast, and greatly reduces eye strain.

For the determination of particle-size distribution, the microprojector is also employed, but for this purpose a 90X oil-immersion objective and aplanatic condenser are placed on the microscope, and the tube length is adjusted to give a magnification of 10,000 diameters. This permits direct measurement of the images by means of a scale ruled directly on the translucent screen. As determined by this method, from a number of samples, the geometric mean diameter of the dust particles is uniformly close to 0.4 microns.

Standards of permissible dustiness show some variations. One, which has some acceptance, places the permissible limit at 10 millions of particles per cubic foot of air, as determined by the standard impinger technique prescribed by the U. S. Public Health Service. Another regulation places the permissible limit at 15 millions of particles per cubic foot of air when the numerical proportion of free silica particles does not exceed 35 per cent; or a lower limit of 5 millions of particles when all the dust particles are free silica. This regulation, also, is based on determinations made according to the standard impinger technique.

Although the total silica content of the rock in the Butte district often runs as high as 60 per cent, or more, the free silica, or quartz, content is considerably less because much of the total silica is present in combined form in silicates such as the feldspars and micas. Free silica in the dust is also lower than in the unfractured rock because the quartz is harder than the feldspars, and micas, and breaks less readily into small particles. By weight, the proportion of free silica in the dust ranges from 10 to 20 per cent; and the numerical proportion is probably less since many of the quartz particles are the largest present in the dust.

Estimates of the average dust concentrations to which men employed at different places in the mines are exposed are based upon the proportion of time they are engaged in different operations such as drilling, mucking, timbering, etc. Information as to the dust generated during drilling, and other operations, was obtained by averaging the results obtained from a large number of samples taken in different places. From these figures there was then calculated the average daily atmospheric dust concentration for different types of working places. The results, in terms of an average for all mines, are given in the accompanying table.

Estimated Average Daily Atmospheric Dust Concentration in All Mines

Atmospheric Environment	Percentage of Number of Men in Atmosphere	Average Daily Dust Concentration, Millions Particles per Cu. Ft. Air
General mine air	42.42	2.1
Air of stopes	27.84	1.8
Air of working faces on levels	13.45	1.4
Air of raises	7.13	4.2
Air of shaft stations	4.46	1.8
Air of intake shafts and skip-loading stations	2.63	3.1
Air of intake shafts	2.07	0.7

Air Cooling in the Gold Mines on the Rand

By Willis H. Carrier,* Member A.I.M.E.

(New York Meeting, February, 1938)

Particular interest in the ventilation of deep mines, especially those in South Africa, has been created by a very complete system of cooling of the world's deepest mine, the Turf shaft of the Robinson Deep in Johannesburg. The writer visited this mine in January, 1936, and again in 1937. During these visits and subsequently, a considerable part of the theory involved in rock cooling in tunnels was developed and this has been checked by practical observations of the rates of rock cooling made within mines. The purpose of this paper is to set forth the method of attack of the mine problem.

The cooling or air conditioning of mines by the use of mechanical refrigeration was not wholly new[1–5] when the Robinson Deep installation was made. The first mine to be cooled was the Morro Velho mine, in Brazil. The first installation in this mine was put into operation in December 1920, the refrigerant being ammonia.[5] The surface cooling gave considerable benefit but it proved inadequate as greater depths were reached, therefore in 1929 two centrifugal refrigerating machines were installed at the 6000-ft. level. These were used in conjunction with the surface cooling system to furnish additional cooling at the lower levels. This later installation used Carrene as a refrigerant. This nonflammable refrigerant, having a boiling point of 104° at sea level, insured reasonable safety, since all parts of the machine operated at subatmospheric pressure. Very favorable reports have been received on the effectiveness of this installation, but no recent data are available.

An interesting feature of that plant is that the condenser water is cooled by spraying it into the exhaust air from the mine at the base of the ventilation upcast, which permits the water to be cooled within a few degrees of the wet-bulb temperature of the air entering the upcast, and the only water that has to be supplied for condensing purposes is that lost by evaporation, or about 0.5 per cent of the volume circulating through the condenser. This plan affords an ideal means for the removal of the heat load from the refrigerating system, and of course also from the mine itself. In many mines, however, especially those in South Africa, there is no

---

Manuscript received at the office of the Institute April 27, 1938. Issued as T.P. 970 in Mining Technology, September 1938.

* Chairman of the Board, Carrier Corporation, Syracuse, N. Y.

[1] References are at the end of the paper.

separate ventilation shaft available, and either mine water has to be employed and then pumped to the surface or condensing water has to be provided through pipes carried down the mine shaft and then pumped back. The latter method is rather expensive on account of pumping cost

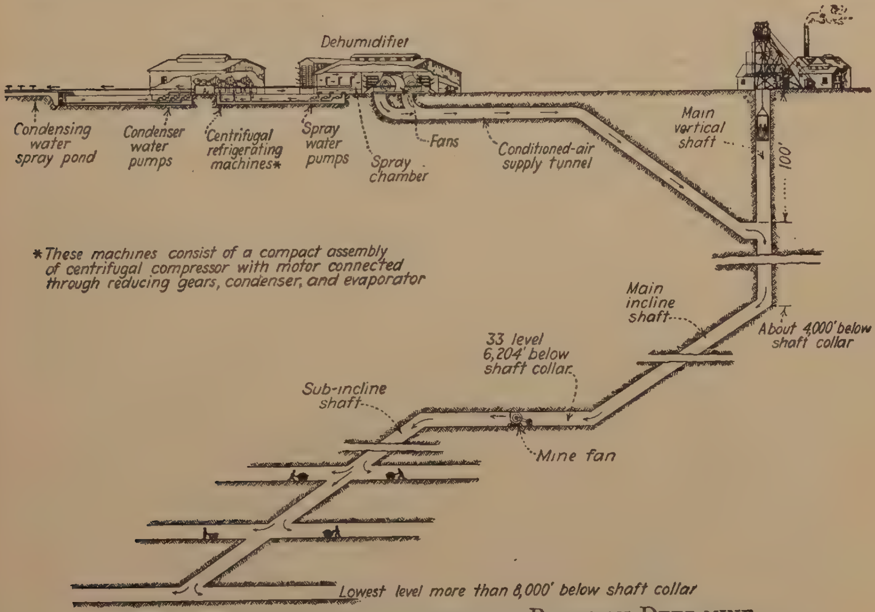

Dehumidifier
Condensing water spray pond
Condenser water pumps
Centrifugal refrigerating machines*
Spray water pumps
Spray chamber
Fans
Conditioned-air supply tunnel
Main vertical shaft
100'
*These machines consist of a compact assembly of centrifugal compressor with motor connected through reducing gears, condenser, and evaporator
Main incline shaft
About 4,000' below shaft collar
33 level 6,204' below shaft collar
Sub-incline shaft
Mine fan
Lowest level more than 8,000' below shaft collar

FIG. 1.—DIAGRAM OF AIR COOLING AT ROBINSON DEEP MINE.

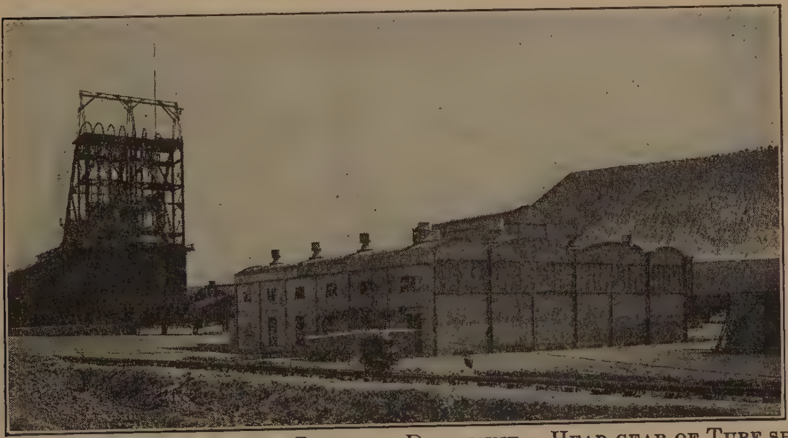


FIG. 2.—AIR-COOLING PLANT AT ROBINSON DEEP MINE. HEAD GEAR OF TURF SHAFT AT LEFT.

and piping. At the Robinson Deep mine no upcast shaft is available and after carefully weighing the advantages and disadvantages of surface versus the underground system, the former was chosen on the basis of practical considerations (Fig. 1).

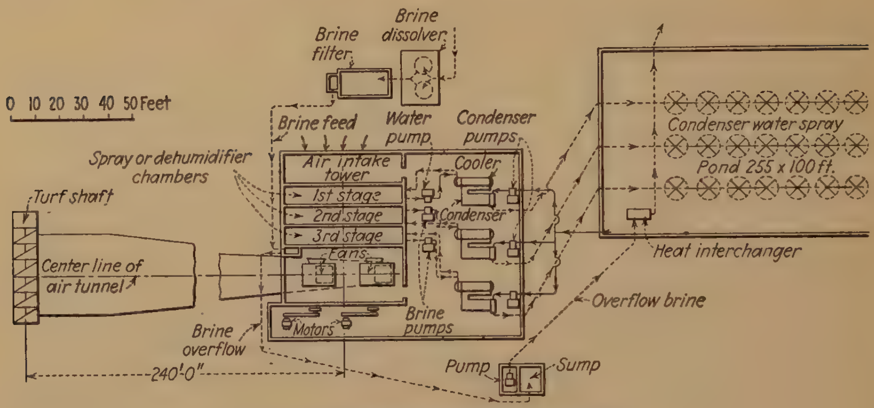


FIG. 3.—THREE-STAGE DEHUMIDIFIER.

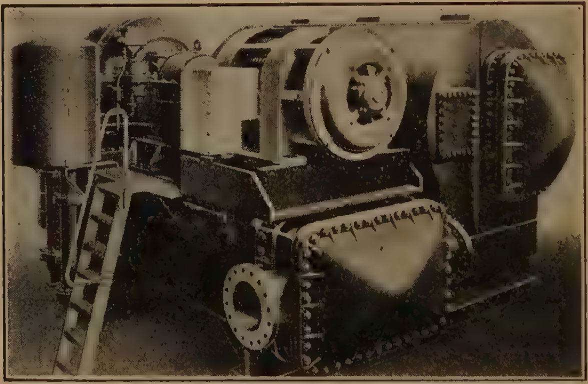


FIG. 4.—CARRIER CENTRIFUGAL REFRIGERATING PLANT.

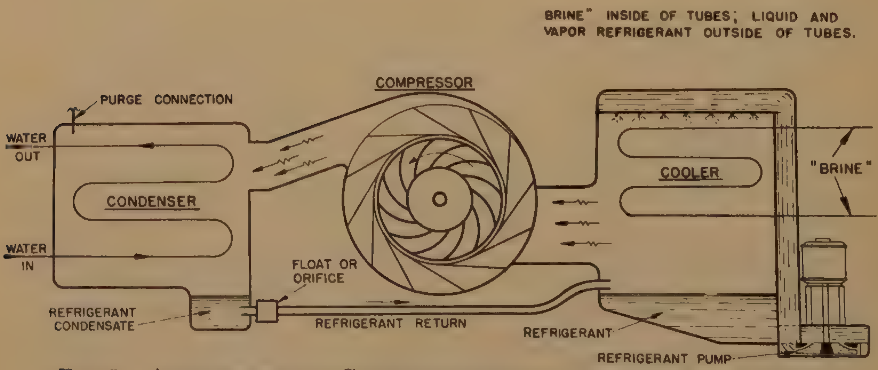


FIG. 5.—ARRANGEMENT OF CARRIER CENTRIFUGAL REFRIGERATION SYSTEM.

This plant (Fig. 2) supplies approximately 360,000 cu. ft. of air per minute, cooled to 33°. The air is introduced 110 ft. below the collar of the shaft, at a balanced pressure, so that there is only a slight outflow at the surface. This control is effected manually by speed control of the air-conditioning fans. The plant was designed to cool 407,000 cu. ft. of air per minute from the maximum wet-bulb temperature of 65° to 38°. Actually, it cools the smaller quantity of air from 65° to 33½°. This requires approximately 2100 tons of refrigeration at the 5600-ft. surface elevation of the mine. The air is cooled by passing through a three-stage horizontal spray-type dehumidifier (Fig. 3). The first stage employs water cooled to about 40°. The second and third stages are supplied with brine. Each stage has its own independent cooling system comprising a complete centrifugal refrigerating system (Figs. 4 and 5). The refrigerant employed is Carrene No. 2 (monofluorotrichloromethane) which has a boiling point at sea level of approximately 75° F. The condenser water is cooled by means of a spray-type cooling pond. The various temperatures involved in actual operation are best indicated by Table 1, taken from recent logs under maximum working conditions.*

TABLE 1.—*Typical Temperature Log, Robinson Deep Mine Plant, under Maximum Load Conditions*

READINGS TAKEN DEC. 28, 1937

Time	Air Temperatures, Deg. F.					Temperatures of Chilled Water and Brine, Deg. F.						Temperatures of Condenser Water, Deg. F.					
	Outside		Dewpoint			Unit 1		Unit 2		Unit 3		Unit 1		Unit 2		Unit 3	
						Water		Brine		Brine		In	Out	In	Out	In	Out
	Dry Bulb	Wet Bulb	Stage 1	Stage 2	Stage 3	In	Out	In	Out	In	Out						
10 a.m.	74	64	52.5	43.	33.	51.5	43.	41.	34.5	33.	28.	72.	80.	71.5	79.	72.	78.
11 a.m.	76	64	53.	43.	32.5	52.	43.	41.5	35.0	33.	27.5	72.5	80.5	72.	80.	73.	79.
12 a.m.	77	64	53.	44.	33.5	52.	44.	42.	35.5	33.5	28.5	73.	81.5	73.	80.5	73.5	80.
1 p.m.	81	65	54.	44.5	33.5	53.	45.	42.5	36.	34.	29.	74.5	82.5	74.	81.5	74.5	81.
2 p.m.	81	65	54.	44.5	33.5	53.	44.5	42.5	36.	34.	29.	74.5	82.5	74.	81.5	74.5	81.
3 p.m.	80	64	52.	43.	32.0	51.	43.	41.	35.	33.	28.	74.0	81.0	73.	81.0	74.	80.

	Unit 1 First Stage	Unit 2 Second Stage	Unit 3 Third Stage
Maximum refrigeration tonnage	800	630	600
Corresponding horsepower	670	625	670

---

* The author is indebted to the paper by K. J. MacWilliams and J. T. McIntyre[6] for the data and curves referring to the rock temperatures in the Robinson Deep.

The results in the mine have been somewhat better than anticipated. Previous to the installation of the cooling equipment, there had been at the bottom of the mine shaft a variation in wet-bulb temperature between winter and summer of about 6°. During its first summer of operation air conditioning prevented this normal rise of 6°. This was no inconsiderable gain (Fig. 6) when it is considered that the maximum wet-bulb temperature in the main airway at the 8000-ft. level was 85°, and that heat strokes were of frequent occurrence among the miners working in the stopes. Subsequently the wet-bulb temperature has been lowered 8° to 9° below the former maximum. With the equipment now in use and the present mine depths of 8500 ft., the "ceiling" of cooling effect is apparently in the neighborhood of 10° wet bulb.

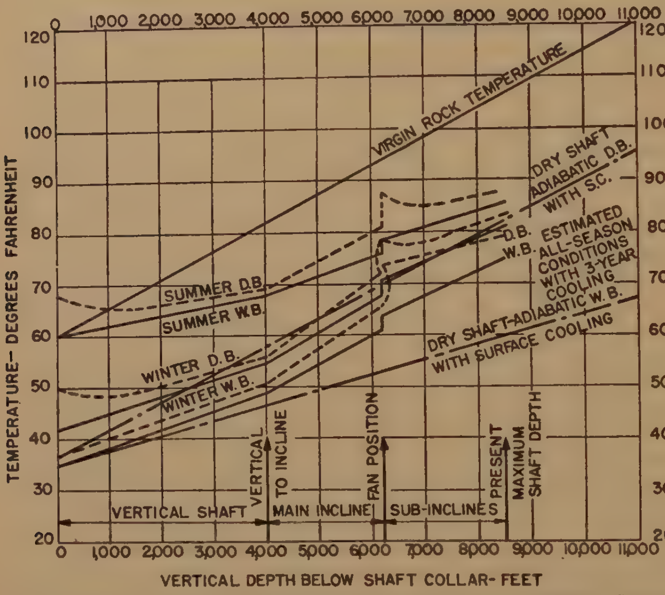


FIG. 6.—DOWNCAST AIR TEMPERATURES, TURF SHAFT, ROBINSON DEEP MINE.

Pneumonia is a frequent occurrence among the miners on the Rand, especially at the deeper mines. Much of this is undoubtedly caused by going from an excessively hot and humid atmosphere to the colder atmosphere at the surface. It was predicted by many mine managers that the introduction of 33° air at the top of the shaft would be a serious additional hazard. That this fear is entirely unfounded is proved by the statistics at the Robinson Deep since the cooling was installed. Not only was the incidence of pneumonia not increased, but, on the contrary, it was decreased 31 per cent. This indicates very satisfactory plant operation.

From casual consideration, it might appear that a great deal of cooling is required at the surface to produce a small cooling at the bottom of the mine. This, however, does not indicate inefficiency. Owing to the great amount of latent heat in the air at high wet-bulb temperatures, it will be found, for instance, that at constant barometric pressure three times as many heat units per pound of dry air must be abstracted to lower the temperature 1° wet bulb at 90° wet bulb as it does to lower it 1° wet bulb at a 40° wet-bulb temperature level. In addition to the great difference in temperature range required at the surface, as compared with the corresponding temperature range at the bottom of the mine at the higher temperature level, there are several other factors to consider. First, there is the heat of adiabatic compression of the air due to the work done upon it by gravity in the air column of the shaft. One British thermal unit is added to each pound of air for every 778 ft. of decrease in elevation. This would give a 1° rise in temperature for every 187 ft. if no moisture were added to the air; second, there is the heat that is added by the cooling of the surrounding rocks; and third, there is the addition of heat from workmen, from electric power, and from oxidation processes. All of these constitute the heat load that must be taken care of by ventilation.

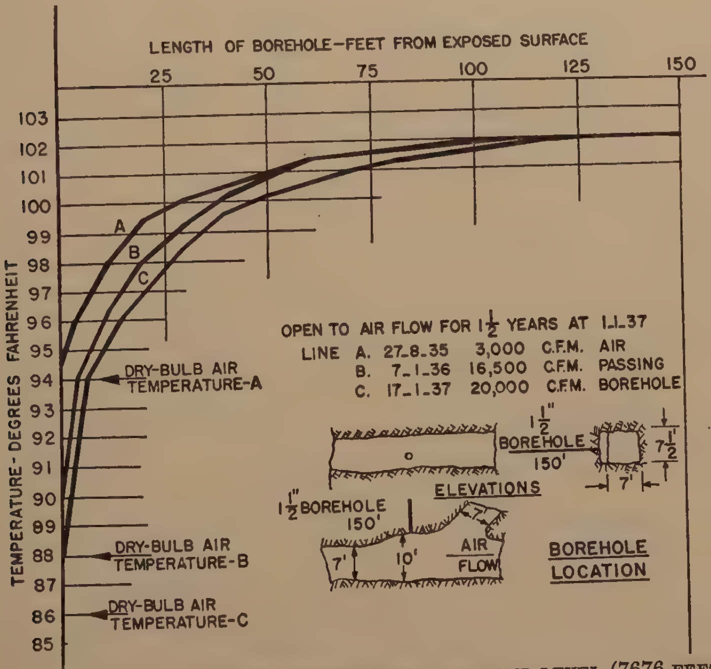


FIG. 7.—BOREHOLE ROCK TEMPERATURES AT FORTY-SECOND LEVEL (7676 FEET BELOW COLLAR).

That the flow of heat from the rock is of some magnitude is best indicated by temperatures taken at various depths in test boreholes at the Robinson Deep. At the 42d-level test borehole, 7676 ft. below the collar, the surface temperature of the rock before artificial cooling was at about 94½°, while 25 ft. into the rock it was nearly 100°. The virgin rock temperature at this mine depth apparently was about 102°, as indicated by the readings at 150-ft. depth. This temperature gradient is best illustrated by Fig. 7. Line *A* shows the temperature gradient in the rock in 1935 before cooling; *B* and *C* are the temperature gradients after cooling. At present the surface temperature is being maintained at about 14° below the virgin rock temperature, but as the surrounding rock is gradually cooled the rate of heat flow diminishes and the rock then becomes an insulator. Fig. 8 shows the temperature gradient existing after 25 years of cooling by ventilation (curve *A*). Curve *B* shows the increased temperature gradient due to subsequent artificial cooling one year later.

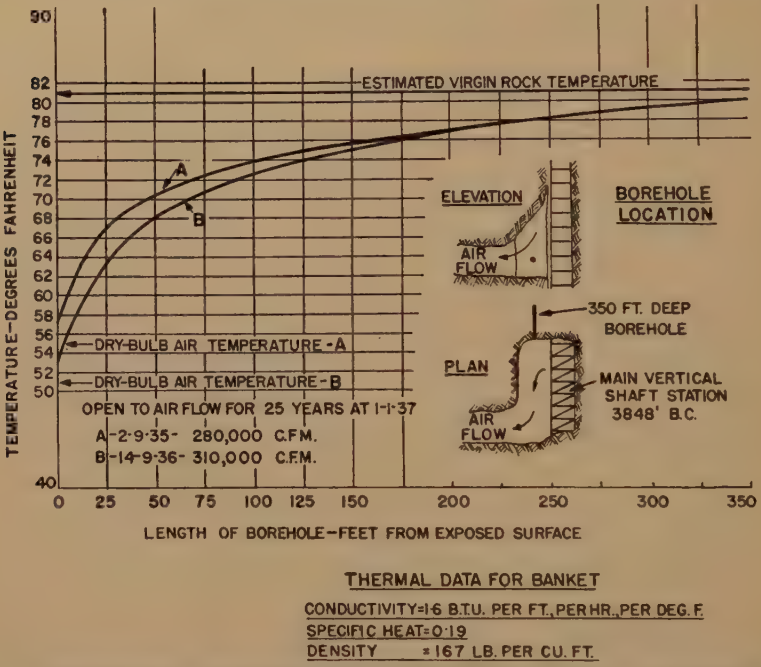


FIG. 8.—BOREHOLE ROCK TEMPERATURES AT SEVENTEENTH LEVEL (3848 FEET BELOW COLLAR).

CALCULATION OF HEAT LOADS

A thorough study of the laws of heat flow into the mine drifts from the surrounding rock has recently been made. By the author's request, contributions to this study have been made by Percy Nicholls and associates in the U. S. Bureau of Mines, and by Dr. Lloyd P. Smith, of Cornell

University, who solved the very difficult mathematical problem of integration necessary to make this calculation.[7] These data have been computed and arranged under the direction of the author into tables and charts for convenience and accuracy of calculation. Such a chart for heat flow in mines is shown in Fig. 9. This chart permits the accurate determination of the rate at which heat flows through any type of rock for any difference between virgin rock temperature and shaft temperature, and also for different periods of cooling from a fraction of a year to many years. These data also make it now possible to calculate with reasonable accuracy not only the present heat load in a mine due to rock cooling but also the increase of that load due to artificial cooling. These calculations when applied to existing mines have been found to be consistent with the observed temperature data, and therefore may be employed with confidence in calculating rock-cooling loads. The author plans to describe their derivation and application in detail in a later paper. Other loads due to electric power and men are easily calculated.

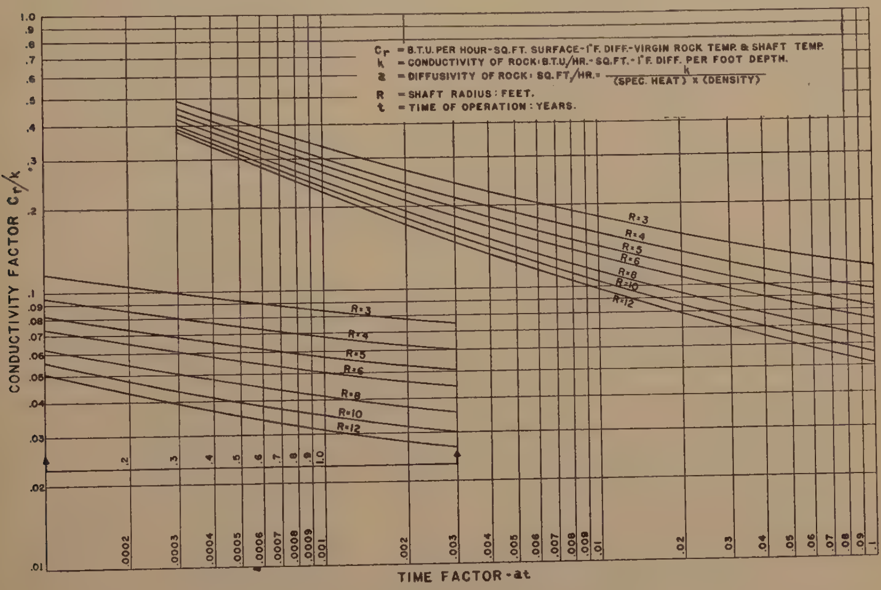


FIG. 9.—CHART OF HEAT FLOW IN MINES.

MEASUREMENTS OF HEAT LOADS

Existing heat loads of this character in a mine may not only be estimated but accurately measured by a very simple method, which may be used as a direct check on the calculated values.

The *wet-bulb* temperature is the one of sole importance in measuring heat added to the air. In a paper written by the author in 1911,[1] it was shown that the wet-bulb temperature alone gives a measure of the heat content of the air, and that this is almost exactly true regardless of the degree of saturation. Therefore, in a path of air flow at constant barometric pressure, wherever heat is added, or removed, to determine accurately the amount of heat change it is only necessary to take the initial and final wet-bulb temperatures between these points. In mining calculations, however, the problem is not so simple, because not only is heat inevitably added by adiabatic compression but the *psychrometric characteristics* of the air vary with the barometric pressure at different levels. To make this principle applicable to mine conditions, a very simple chart has been prepared (Fig. 10) which indicates the variations in wet-bulb temperatures occurring at different depths where no external heat is added or removed. One set of curves shows the variation in heat content of the air at different levels with change of wet-bulb temperature, while the other set of curves shows the variation both in temperature and

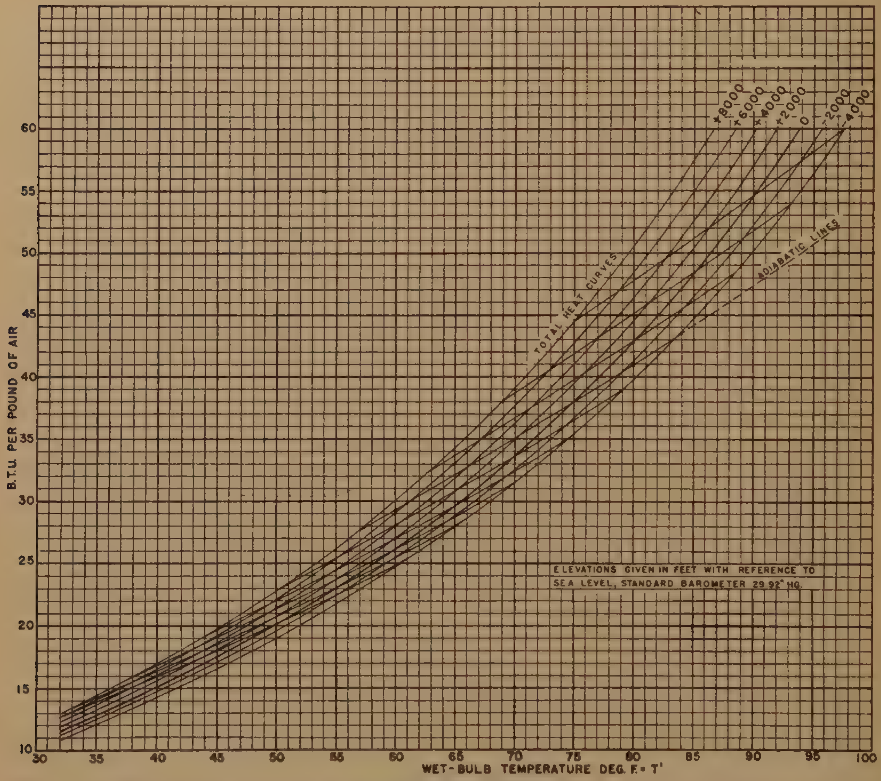


FIG. 10.—VARIATIONS IN WET-BULB TEMPERATURES WHEN NO EXTERNAL HEAT IS ADDED OR REMOVED.

Based on assumed partial saturation of air to 80 per cent relative humidity.

heat content where there is no addition of heat except that due to adiabatic compression. In other words, this second series of curves represents true adiabatic wet-bulb temperatures for different levels. Tables of temperature variation of a similar character have been given by J. S. Jones[3] and Professor Jeppe, of the Union of South Africa. Fig. 10, however, has several advantages in that it gives an infinite selection of temperatures and levels with a high degree of accuracy and also, most important of all, gives the corresponding heat contents.

The great serviceability of such a chart will be appreciated when it is realized that by taking the wet-bulb temperatures at any two different levels in the mine it is possible to place them on the chart and find exactly how much heat has been added between the points. The practical value of this is that it permits a ready determination of just how much additional cooling can be secured by increase of natural ventilation, or by lowering of air temperature. Consider the practical example at the Robinson Deep. Here the average annual wet-bulb temperature at the collar of the shaft before the installation of the air-conditioning system was 53°. The average wet-bulb temperature at the 6000-ft. level, to which the air is drawn by the ventilation fan, was 71°. The corresponding adiabatic wet-bulb temperature from the chart is 67°. The increase in wet-bulb temperature due to external sources was, therefore, 4°, and the corresponding thermal unit difference as taken directly from the chart is 3.2 B.t.u. per pound of air. Since in this passageway there are 21,000 lb. of air handled per minute, the total heat from the rock and other sources between these points was equivalent to 305 tons of refrigeration.

This discussion inevitably will raise a question in the mind of the engineer as to the relative merits of surface cooling and underground cooling. Calculations made by the author on the Robinson Deep, from observed temperatures, show that after three years of operation the increase of heat load from the increased rock cooling is a relatively small percentage of the total heat load. In other words, a plant of the same average annual capacity placed at the bottom of the mine would have shown but slight advantage in cooling effect over that of a surface plant. However, the surface plant has to be sufficiently large to take care of the peak loads occurring at the surface. The variation, therefore, is from 100 per cent capacity to zero capacity when the temperature outside is below 32°, and the plant is not in operation. The average annual load for this locality is about 55 per cent of the peak capacity. This would permit a much smaller equipment to be used if the cooling were done by an underground plant. The latter would also have the advantage of operation under continuous full load, which, as will be appreciated, has a great operating advantage. The underground plant would work at relatively high temperature levels, both on the evaporator and the condenser side. This gives greatly increased compressor capacity but does not materially change the amount of annual power required. To obtain the full advantages and economy of an underground plant would require, however, at the Robinson Deep the sinking of a 6000 or 7000-ft. shaft for ventilation purposes only. Such a shaft to handle the full quantity of air (407,000 cu. ft. a minute) would have cost more than the increased cost of the refrigeration plant at the surface. It would seem, therefore, from the economic standpoint in connection with the Robinson Deep that the owning and operating cost of the larger plant at the surface was warranted. There is, however, one other point that has to be given consideration, which is entirely in favor of the underground plant. For practical reasons, the limit of cooling at the surface was selected at 33°. The mean annual wet-bulb temperature is about 52½° at Johannesburg, so that the average cooling effect at the surface is about 19½° wet-bulb temperature, which will give a definitely limited cooling effect at the bottom of the mine, as indicated by Table 1.

The ultimate average cooling effect available at the 8500-ft. level therefore is only about 8½° wet bulb with surface refrigeration. This will permit the extending of the workings at least 1500 ft. below the present maximum depth of 8500 ft. With an underground plant, any desired cooling effect could be obtained at any depth. The practicable depth of mining would then depend entirely on practical questions of rock pressure and cost of ore handling. In mines that have a very high rock temperature—for example, the Magma mine at Superior, Ariz.—an underground installation becomes far more practicable because of the large increment of heat that will be added from the surface to the bottom of the mine and also because of the much lower cost of a separate ventilation shaft. The great differences in rock-temperature gradients found in the Magma mine as contrasted with those in the Rand mines is shown by comparison of the temperatures at different levels. In the Robinson Deep mine, the virgin rock temperature at 7800 ft. below the collar has been measured at approximately 103°, while at the Magma mine the virgin rock temperature is said to be about 140° at 4000 ft. In the Rand mines of South Africa, owing to favorable rock temperatures, the limit of mining is about 8500 ft. without artificial cooling. In mines like the Magma, artificial cooling becomes necessary below the 3500-ft. level and advantageous at the 3000-ft. level. It is believed that artificial mine cooling, once its worth is proved, will become much more generally used than at present in hot mines. It can easily be shown that the owning and operating cost of an artificial cooling plant may be considerably less than the owning and operating cost of additional ventilation and ventilation shafts to accomplish the same results by natural means. In other words, when the normal requirements of ventilation have to be increased considerably on account of heat load from the rock, it becomes cheaper to produce the additional cooling artificially rather than by additional ventilation.

REFERENCES

1. W. H. Carrier: Rational Psychrometric Formulae. *Jnl.* Amer. Soc. Mech. Engrs. (1911) **33,** 1311.
2. T. T. Read and F. C. Houghten: Cooling of Mine Air. U. S. Bur. Mines *R.I.* 2554 (1923). Refers to 12 previous publications.
3. J. S. Jones: Hygrometry for Deep Mines. *Trans.* Min. and Met. Soc. (1924–1925).
4. W. H. Carrier and C. O. Mackey: A Review of Existing Psychrometric Data in Relation to Practical Engineering Problems. *Trans.* Amer. Soc. Mech. Engrs. (1937) **59,** 33.
5. Air-Cooling Plant at the Morro Velho Mine of the St. John del Rey Mining Co. Ltd., Brazil. *Trans.* Inst. Min. Engrs. (1922) **63,** 326–341, 424–427.
6. K. J. MacWilliams and J. T. McIntyre: Problems Arising Out of Temperature and Humidity in Deep Mining on the Witwatersrand—Surface Refrigeration at the Turf Shaft of the Robinson Deep, Limited. *Jnl.* Chem., Met. and Min. Soc. South Africa (March 1937).
7. L. P. Smith: Heat Flow in an Infinite Solid Bounded Internally by a Cylinder. *Jnl. Applied Physics* (June 1937) **8.**

DISCUSSION

J. T. McIntyre,* Johannesburg, Transvaal, South Africa.—The author has summarized almost too briefly some of the difficulties of air cooling in deep mines. The work to date has certainly indicated the most useful possibilities for plant of this type in the ventilation of deep and hot mines, but much remains to be done on the mining side in providing suitable airways for conveying the ventilating air, whether cooled or not, to the working zones. As a guide to the mining man on the importance of dry airways, Dr. Carrier might have stressed the importance of maintaining "spread" between wet-bulb and dry-bulb air temperatures. The point is made in the paper that the wet-bulb temperature is the most important as far as heat content of the mine-ventilating air and cooling of the mine workings is concerned, but it should be noted that the dry-bulb temperature is the real criterion for heat flow from rock to air. Thus a horizontal ventilation split from the main downcast airway going 3000 ft. to the boundary gives relatively low wet-bulb temperatures at the boundary, provided that no moisture is picked up in the split and provided, also, that the dry-bulb temperature is about equal to the virgin rock temperature at entry. With open drains and a wet footwall from dripping cars, however, the wet-bulb temperature rises to within a few degrees of the virgin rock temperature at the boundary. "Dry" mining, therefore, is a most important essential for air conditioning in hot deep mines, with or without cooling plants of the type described.

On the Witwatersrand the dangerous silicotic dust and the present necessity for wetting down everywhere adds to the difficulties of deriving the fullest benefits from air cooling, and is an additional important consideration.

---

* Consulting Engineer's Department, New Consolidated Gold Fields, Limited.

Suggestions for the Control of Silicosis in Mining

By Donald E. Cummings*

(New York Meeting, February, 1938)

Measures appropriate for the control of the silicosis hazard in mining cannot be formulated precisely, but sufficient knowledge[1-19] has accumulated during the past quarter century to permit the suggestion of useful, though partly empirical, rules for its alleviation. The industrial hygienist today recognizes that the peril incurred by the inhalation of harmful dust is *a function of two variable factors—the degree of harmful exposure and the specific susceptibility of the exposed individual to pulmonary injury.* Control of the danger arising out of prolonged contact with pernicious airborne particulate matter is consequently best exercised by influencing both of these recognized variables. Obviously the problem has two broad aspects—one, medical; the other within the province of the engineer.

I—Factors Comprising Degree of Individual Susceptibility

Thorough investigations conducted among miners who have labored for years in the presence of recognized silicosis hazards usually reveal an impressive difference in the injury incurred by men whose exposure to dust has been nearly identical. After a quarter century underground, one individual is short of breath and incapacitated with miners' complaint, while a fellow workman suffers no pulmonary embarrassment and is in good health. Physicians ascribe this oft-observed difference in reaction to individual susceptibility and express a universal regret when they deplore the fate that chose to make the predisposed youth a miner rather than a farmer. To trust the health of miners to fate at the present time is even more unfortunate, because many of the factors that influence man's susceptibility to dust are known, and can be accurately determined by competent medical methods. Among the most important factors comprising susceptibility to injury by dust are the anatomical peculiarities of the individual's respiratory mechanism, his physiological response to the inhalation of dust, his previous dust exposure, the existence of pulmonary infections or other pathological processes in his lungs, his age,

---

Manuscript received at the office of the Institute Feb. 8, 1938. Issued as T.P. 930 in Mining Technology, May 1938.

* Assistant Director, in charge of field studies, The Saranac Laboratory of the Edward L. Trudeau Foundation, Saranac Lake, New York.

[1] References are at the end of the paper.

nationality, general health, and his ability to cooperate with others in achieving a trustworthy environment.

The upper respiratory tract plays an important part in conditioning the air which miners breathe during their work. G. Lehman[20] demonstrated that the efficiency of nasal filtration in a group of silicotic miners was 40 per cent lower than that in normal miners, and concluded that jobs with a silicosis hazard should be held only by men with good nasal filters. It is obvious that if a large proportion of inhaled dust is retained in the upper respiratory tract, from whence it may be expelled, little will remain to penetrate the deeper tissues in which it may be permanently trapped. C. E. Brown[21] has studied the retention of inhaled dust by man, and his data indicate that approximately 50 per cent of the particulate matter breathed by healthy workmen will be retained throughout the respiratory tract. The proportion of this dust finally trapped in the ultimate pulmonary structures is probably* of the same order as that removed by the cloth filters of a commercial dust collector equipped with a good primary separator, which compares with the upper respiratory tract. Even after dust has entered the lungs of healthy miners there are wide variations in individual tolerance depending upon the size and efficiency of the pulmonary drainage system[22] known as the lymphatics. *Certainly any abnormal peculiarity of the respiratory tract that increases the ingress of air-borne particulate matter into the ultimate pulmonary structures, or which interferes with its proper disposition in the lung, enhances the individual susceptibility to injury by silica dust.*

Each miner's reaction to dust also varies with his physiological peculiarities. The individual whose mucous membranes are moist is surely provided with a better filter than the one whose respiratory tract is dry. Certain men find dusts very irritating when inhaled and manifest their sensitivity by a contraction of the air passages accompanied by a swelling of the lining membranes and an increase of secretion, all of which tend to protect the lung from the irritant. Within the air sacs, dust particles are engulfed by mobile cells, known as phagocytes, which are

---

* If it is assumed that a workman breathes at the rate of 1.0 cu. ft. per minute, and that the limiting concentration of silica that can be breathed without contracting silicosis is 0.02 mg. per cu. ft. of air:

$$120{,}000 \times 0.02 = 2400 \text{ mg. of silica breathed per year.}$$

At 50.0 per cent retention, 1200 mg. will remain in the respiratory tract. At autopsy, the limiting concentration of silica that may be breathed without inducing silicosis appears to be about 1.0 per cent of the dried lung. Assuming that the average lung weighs 200.0 grams when dried, then

$200.0 \times 0.01 = 2.0$ grams, or 2000.0 mg., of silica "safely" retained in a lifetime.
$2000.0 \div 40 = 50.0$ mg. per year may be "safely" retained for 40 yr., etc.
$50.0 \div 1200.0 = 4.0$ per cent ultimate retention.

normally stimulated to rapidly remove irritating substances such as silica into the domain of the lymphatic drainage system. Gardner states[22] that when these phagocytes have already burdened themselves with inert material such as carbon they are unable to respond to stimulating particles in the usual manner. As a consequence the miner who has received no significant injury from prolonged contact with coal dust[7] may not react to a subsequent silica exposure in the same manner as the man with no previous dust exposure. The only unerring method of disclosing an individual's susceptibility to silicosis is to observe his reaction to silica dust under nicely controlled conditions. *Usually any miner who shows measurable pulmonary changes in excess of those noted in the group as a whole, or which presage the development of disability in less than a normal working lifetime, may be considered to have an unfavorable susceptibility or to be working in an unusually hazardous atmosphere. Additional investigations will reveal the relative weight to be accorded each of these possibilities.*

Previous dust exposure quite obviously affects the susceptibility of any miner, since many of the particles inhaled are nearly insoluble, or only slowly soluble, and therefore accumulate until the normal drainage mechanism[22] is disturbed or the physiological function[23] altered so that the limit of tolerance for dust is considerably reduced. The types of air-borne contaminants previously inhaled must be carefully considered, for they may vary all the way from limestone, which dissolves in the lung and is apparently harmless,[24] to quartz, which does not dissolve[25] to any appreciable extent and is specifically injurious. It is desirable to estimate the approximate amount of each type of dust retained in the lung in order to ascertain the individual's capacity for additional exposure and to appraise the degree of any modifying influence[22] which other substances may exert upon the tendency for silicosis to develop. *Analyses*[26,27] *suggest that healthy miners who do not retain more than* 50 *mg. of fine silica particles in their lungs during any year may not incur silicosis in an average working lifetime. This limiting value may be altered, of course, by the presence in the lung of other dusts that tend to modify the characteristic response to silica.*

The layman can appreciate almost as readily as the pathologist[28] the influence that a pre-existing pulmonary injury might exert upon a miner's reaction to a subsequent noxious dust exposure. If a scar has developed in the lung because of a previous pneumonia which failed to resolve completely, or because of an unrecognized tuberculous infection, it is apparent that the scar-forming tendency of fine silica dust will be augmented in those areas, so that pulmonary damage may develop far more readily than it would under normal conditions. In fact, it is quite possible to conceive of a tuberculous individual whose health is so delicately balanced that he will remain well while working in a normal atmosphere but will become ill when exposed to concentrations of pernicious dust which would not seriously affect a healthy man. It is equally apparent that other pathological conditions of the lung, such as asthma, bronchitis, or pleurisy, may be aggravated by dust inhalation or may affect the normal pulmonary reaction sufficiently to hasten the development of disability following dust exposure. *Usually the existence of chronic pulmonary infections, the sequelae from previous pulmonary injuries, or the presence of any marked pathological change in the lung can be accepted as reliable evidence of predisposition to injury by harmful dusts.*

Age is a factor in individual susceptibility for several reasons. The tuberculosis mortality rate[29] among white males tends to increase steadily with age, and since that disease is so inextricably linked with silicosis, it would appear that the hazard from silica should follow a similar course. Moreover, with advancing age the tolerance for dangerous dust becomes reduced[30] because of the material continually accumulating in the lung from natural exposures and because of the changes in the tissues inherent with age. On the other hand, young miners do not always work at a steady pace but frequently overexert themselves and thus tend to increase their susceptibility to infections. Brown and Sampson[31] believe that much of the clinical tuberculosis which develops among later age groups is due to infections contracted in youth. *Generally the tolerance for dangerous dust is reduced as age increases while the susceptibility to chronic infections varies with age and the habits of living and work.*

Nationality plays a role in the susceptibility to injury from dust, not because of racial differences in the reaction to silica, but because certain nationalities are much less resistant to chronic pulmonary infections than others. Negroes, American Indians, and Mexicans[32] all suffer from an unusually high tuberculosis mortality and for that reason constitute a greater hazard when exposed to harmful dust. The Finnish people, large numbers of whom are engaged in the mining industry, have the highest death rate from tuberculosis among the white races. *In general, individual susceptibility to injury from dust is greater among those nationalities with the least resistance to tuberculosis.*

The state of an individual's general health also determines his susceptibility to a dust hazard because that peril is so intimately allied with tuberculosis, which in turn depends on the general well-being. Miserable living conditions associated with a lack of understanding of hygiene or with a poor economic status, chronic infections or debilitating diseases, and unhealthy working environments all tend to undermine the general health and diminish the reserve of resistance to tuberculosis. Unfortunately opportunities for contact with the tubercle bacillus are always greater among the poor and among groups exposed to dust hazards but not under medical control. *The hazard of silicosis is surely amplified in any individual whose general health is impaired or whose likelihood of contracting tuberculosis is augmented.*

The degree of danger present in a hazardous environment can be mitigated by miners who are sufficiently intelligent and willing to combat dust by adopting the procedures and protective apparatus designed for their safety. If they fail to take advantage of such benefits as can be assured by the proper use of water or ventilating equipment, or if they refuse to wear satisfactory respirators, their susceptibility is, of course, increased. *No single factor contributes more to a miner's susceptibility to dust than his inability to cooperate willingly and understandingly in a silicosis prevention program.*

II—FACTORS COMPRISING DEGREE OF EXPOSURE

Investigators who first studied the problem of pulmonary disease among miners were puzzled by the remarkable variation in the hazard associated with different types of mining or with the location of mines engaged in extracting the same mineral. The recovery of gold from the hard rock of the Witwatersrand[33] seemed to involve a greater risk to health than did coal mining in Scotland, but it was also recognized that certain Welsh coal miners had a very bad experience. The whole situation was most confusing. Fortunately many of these enigmas have been solved by patient study, so that today most of the factors which determine the degree of an injurious exposure are clearly recognized. Among them may be listed: the composition, size distribution, physical characteristics and concentration of dusts suspended at the breathing level, together with the duration of exposure and the general conditions of work.

In England, Collis,[34] in the Milroy Lecture for 1915, associated a high mortality among the workmen in certain dusty trades with the inhalation of silica. Since this statistical study, the observation has been confirmed repeatedly by post mortem analyses[26] of affected lungs, by clinical investigations[35] and by experimental reproduction of the disease[36] in laboratory animals. Today free silica is generally acknowledged to be specifically injurious when inhaled in finely comminuted form and great importance is attached to the substance when present in dusts. Moreover, certain forms of silica[37] are more injurious than others though none appears to rival quartz in its ability to injure living tissues. Not only does the silicosis hazard depend upon the presence of free silica and the variety of modifications it may assume but also upon the admixture of other dusts[22] which influence its behavior in the respiratory tract. While there is limited evidence[38] to show that certain materials such as alkalies tend to accentuate the harmful qualities of silica dust, there is an abundance of data to suggest the favorable action of inert substances such as carbon,[7] hematite,[37] alumina,[39] or gypsum[22] upon the injurious property of free silica. *Nevertheless the composition of a mine dust may be regarded as* *potentially hazardous if free silica, in any of its modifications, is found to constitute* 10.0 *per cent or more of a suitable sample of air-borne particulate matter.*

McRae[26] has shown that few particles greater than 10 microns in diameter are found in the ultimate air sacs of the silicotic lung, while Gardner and the author[40] have demonstrated that noxious dusts less than 3 microns in size are particularly harmful. No one has accurately determined the lower limit of particle size capable of producing silicosis—if there is such a limit—but it is believed that most of the damage to a silicotic lung is undoubtedly produced by the principal mass of retained particles composed of dusts greater than 0.5 micron in diameter. *If a mine dust possesses qualities that make it potentially hazardous, the size fractions that may be considered physiologically significant are those between* 10.0 *and* 0.5 *microns in diameter.*

The physical characteristics of a mine dust deserve careful evaluation, for though the hardness and sharpness[41] of very fine particles are not associated with their capacity for inflicting damage to the lung, other properties are known to influence this attribute. The tendency for certain finely divided materials to form aggregates because of opposite electrical charges, or because of an affinity for moisture, sometimes reduces the magnitude of a hazard by simultaneously increasing the apparent diameters of suspended dusts above the size range most readily inhaled while decreasing the total number of particles in the atmosphere. The behavior of dust suspensions may vary with the relative humidity, for the ability of certain components to be more readily wet than others hastens their selective removal[16] from a saturated atmosphere. If a mine dust is composed of substances with widely divergent specific gravities, the heavier constituents will tend to settle out more rapidly than the lighter, thus changing the nature of the air-borne contaminants where an opportunity for such settlement is afforded. Since the minute particles involved in dust diseases possess a large surface area per unit of mass, all the phenomena that depend on surface characteristics assume great importance, including the propensity for adsorption or solubility to occur. In fact, it appears reasonable to suppose that the deleterious effects produced in the lung by fine dust particles may be due to the activity of their surfaces, and that opportunities for diminishing a hazard offered by a fixed exposure to silica must be largely confined to procedures that will satisfy this specific activity prior to the time when the dust acts upon the pulmonary tissues and injures them. Among the investigations testing the ability of physiologically inert materials to protect the lung from damage by silica surfaces, those in Canadian mines[42] utilizing aluminum dust and experimental studies[37,43] employing iron salts appear interesting. The solubility[25] of fine silica particles in water or body fluids is very small and does not account for the damage which this substance is capable of inflicting, since the most insoluble modification, quartz, is also the most active, while one of the most soluble solid forms, freshly precipitated moist amorphous silica,[37] is apparently quite inert. In a saturated mine atmosphere, however, the solubility of certain common finely divided rock dusts such as limestone, when present with silica, may frequently be of sufficient importance to require evaluation. *In general, the potentially hazardous physical characteristics inherent in a mixed dust are those which tend to maintain the silica particles highly dispersed with active surfaces capable of entering and acting upon the ultimate pulmonary structures. On the contrary, those characteristics of a dust are protective which aggregate fine silica particles or which inactivate their surfaces and impair their ability to penetrate into the deeper portions of the lung.*

The concentration of pernicious dust in a mine atmosphere constitutes a measure of the degree of exposure, but the relationship is not a simple one. During the course of his evolution man has developed a fairly efficient protective mechanism against the air-borne contaminants which have always enveloped the earth, so that he has a considerable tolerance for industrial dusts. Experience indicates that a working lifetime of exposure to free silica is not hazardous to men with healthy lungs if the concentration of particles between 10.0 and 0.5 microns does not exceed a limiting value, which I have designated[44] as the *Primary Threshold.* Healthy miners working in concentrations of silica dust greater than this threshold value tend to develop silicosis after intervals which shorten as the exposure is intensified until finally another value for concentration is reached beyond which silicosis and disability are inevitably contracted after an exposure of a few years. This value I have designated as the *Secondary Threshold.* These thresholds may be altered by the coexistence of other dusts which modify the degree of a potential hazard. *In general, a mine atmosphere is potentially dangerous at any point where the concentration of silica particles between* 0.5 *and* 10.0 *microns in size exceeds* 15,000,000 *to* 20,000,000 (*dark-field count*) *particles per cubic foot of air* (*or* 0.02 *mg. of silica per cubic foot of air*).

Measurements of the duration of exposure to a potential dust hazard are difficult and elusive because of the ever-changing conditions which have to be taken into consideration. Doubtless the miner who occasionally spends a few minutes drilling a dry hole in a quartz boulder with a jack hammer subjects himself to a greater risk than he does throughout the rest of the day when drilling with a good wet drifter in an adequately ventilated rock heading. Everyone interested in silicosis must constantly bear in mind that while the hazard is small with long exposures to low concentrations of silica and remains moderate even at average concentrations, it finally becomes immeasureably great at high concentrations, as has been so strikingly illustrated among sand pulverizers,[38] sand blasters,[45] and miners[46] operating dry drills in siliceous rock. All too frequently an analysis of the degree of an individual's exposure considers only the length of time spent in an average occupational environment, and fails to evaluate the significance of the short exposures to severe hazards which may account for most of the harm incurred. *As a broad generality it may be assumed that men with healthy lungs may work in silica without great danger for 40 years at concentrations not exceeding the Primary Threshold or for 10 to 25 years in concentrations between the Primary and Secondary Thresholds while 2 to 5 years exposure will be dangerous if the concentration is above the Secondary Threshold.*

Mine atmospheres deficient in oxygen or containing an excess of carbon dioxide or other gases produced by the firing of explosives, or which are uncomfortably hot or humid, all tend to increase the degree of the exposure by accelerating the rate of respiration as well as by increasing individual susceptibility. *Any general environmental factor tends to aggravate the degree of exposure if it impairs the efficiency or health of the miners and furnishes no compensating protection against dusts.*

III—Silicosis Control Objectives

To be truly effective, then, silicosis control in mining should be accomplished by combining medical management of the employees with engineering supervision of the dust exposure. If an attempt were made to eliminate all danger from mine dusts without considering the miners' physical condition, it would be necessary to furnish air of about the same quality and in much the same great abundance as found on surface. Though it is conceivable that a mine atmosphere might be made to reproduce surface conditions, it seems impractical to resort to the costly equipment and methods which would be required to achieve such a result. Neither would medical procedures alone prove feasible though it might be possible, by carefully selecting healthy workmen and employing them only so long as they exhibit a tolerance for harmful dust, to avoid the development of disabling disease even in the presence of excessive concentrations of this pernicious material. In the same degree that engineers would find it impractical to maintain surface atmospheres constantly at all underground working places, physicians would find it impossible to provide workmen whose lungs would remain continually healthy and free from infections.

The troublesome problem is to decide what constitutes the optimum degree of medical or engineering control compatible with reasonable operating methods and costs. Though sufficient data are not yet available with which to plot accurate curves representing the value of the hazard for (1) various states of individual susceptibility, or (2) different degrees of injurious exposure, it appears possible that both these graphs might assume a form somewhat similar to that of an exponential function, or might become straight lines if plotted on logarithmic paper. In any event, it is certain that only a moderate improvement in an excessive degree of either individual susceptibility or injurious exposure results in a very appreciable abatement of the hazard. Consequently, efforts expended on the alleviation of the dangers due to dust should always be directed toward the most susceptible miners and toward those operations producing the severest exposures.

IV—Suggestions for Determining the Existence of a Silicosis Hazard

If there is any reasonable doubt about the existence of a silicosis hazard at a mine, it may be dispelled by conducting a preliminary survey[3] to establish reliable information on this point before procedures are designed for the control of the danger. The presence of a hazard may be regarded as proved if any post-mortem examination by a competent pathologist has disclosed nodular silicosis in a miner whose exposure to silica dust was confined to the mine in question. Conclusive proof of a hazard may also be assumed if more than one miner has been found with uniformly distributed discrete nodular shadows in good stereoscopic chest roentgenograms, providing the dust exposure had been confined to the mine under study. There is strong presumptive evidence[29] of a hazard if there have been more than 10 deaths from pulmonary tuberculosis among 1000 miners between the ages of 15 and 64 for the 11 years from 1925 to 1935 inclusive. A silicosis hazard may also be suspected if several miners have become disabled by "miners' complaint" or "miners' consumption" after 20 to 25 years work in the mine under investigation. A dust hazard is also suggested if free silica particles are consistently present in concentrations exceeding 15 to 20 million particles between 0.5 and 10.0 microns in size per cubic foot of mine air surrounding typical working places. (Dark-field counting required.)

V—Suggested Medical Control Procedures for Silicosis Hazards in Mines

A competent mine physician, utilizing the existing knowledge of those conditions which influence an individual's susceptibility, can reduce the silicosis hazard through the proper selection and placement of employees. Applicants for work, as well as regular employees, should be given thorough physical examinations. Each miner's family and previous medical history, together with his occupational record,[44] should be carefully elicited. If the questioner has patience, ability, and experience, he may thus obtain an excellent understanding of each examinee's previous dust exposure and other predisposing characteristics. The knowledge derived from this source cannot be gained by any other means for, it should be noted, both silicosis and tuberculosis are slowly progressive conditions, and certain miners may have already received injurious exposures which have not yet completed their reaction and are, therefore, not fully disclosed by physical examination. Certain clinical laboratory tests, including the enumeration and differentiation of the blood corpuscles, the determination of the rate of sedimentation of the blood and examination of the sputum, are also useful in evaluating individual susceptibility to injury by dust. Stereoscopic chest roentgenograms are essential and should be of good quality.[47] The shadows cast by the lung, when interposed between a sensitized film and a source of X-rays, are both intricate and delicate, and require skillful interpretation. They should be classified[48] according to the degree of reaction resulting from previous dust exposures and according to the evidence indicating the existence of pulmonary infections or other abnormalities. By correlating this evidence the physician is able to arrive at a final conclusion concerning each individual's susceptibility to injury by exposure to harmful dust.

Applicants for Work

In the interest of their future well-being and that of their fellow workmen, applicants for work in a potentially hazardous mine atmosphere should be disapproved if they have a marked predisposition to disability from dust exposure. Unless such individuals are a menace to the public health or physically incapable of the tasks required, they may be approved for nonhazardous occupations. The common causes for disapproval of applicants include active tuberculosis in any form, or chronic pulmonary infections such as unresolved or organizing pneumonia and bronchitis. Among the common pulmonary abnormalities which may alter the reaction of the lung to a subsequent dust exposure are asthma, dilated air sacs (emphysema), massive collapse of the air sacs (atalectasis), air in the pleural space (pneumo-thorax), tumors, pleurisy, and massive scarring due to obstruction of the blood supply, infections, trauma, or reaction to dust. Evidence of a rapid or excessive reaction to a previous dust exposure may also constitute a fair basis for disapproval, though cases of uncomplicated silicosis which have been acquired only after greater than average exposures may be looked upon as good risks and approved for employment in moderate dust concentrations.

The predisposition to tuberculosis recognized among persons with diseases like diabetes, or among susceptible races such as the Negro, American Indian, or Mexican, should be considered carefully before approving this type of applicant for exposure to silica. It is also unwise to select a group of miners composed predominantly of any one race or age group, since the risk can be better apportioned by maintaining an average distribution of nationalities and ages. The presence of marked secondary anemia, malnutrition, low blood pressure, serious sinus disease, unfavorable family history, or direct contact with an "open" case of tuberculosis suggests poor general health and increased susceptibility to pulmonary disease, and tends to increase the danger of silica exposure for an individual affected with any of these conditions. The mental fitness of the applicant is also important, since upon this quality rests his ability willingly and intelligently to follow reasonable instructions for the control of the dust hazard.

Regular Employees

Regular employees should be assured that physical examinations are not to be used to discriminate against them, but rather to place them in occupations or situations best adapted to their physical requirements. In general, these examinations will serve to place miners in one of four classifications: (1) those with healthy lungs, (2) those with uncomplicated nodular silicosis, (3) those with predisposition to pulmonary injury by silica dust, (4) those with evidence of silicosis complicated with tuberculosis or other pulmonary abnormalities.

Men with healthy lungs may be approved for occupations offering exposure to silica. They should report to the medical department after any illness necessitating a week's absence from work, and for complete re-examination annually.

Men with uncomplicated nodular silicosis seldom show evidence of pulmonary disability[44] and, unless they appear to have a condition that is progressing with unusual rapidity, may be approved for occupations with a moderate silica exposure. They should report to the medical department after any illness involving three or more days' loss of work, and for a single chest roentgenogram every six months as well as for complete annual re-examination.

Miners discovered to possess a marked predisposition to injury by dust should not be approved for occupations involving that hazard. The physician should recommend men in this group for hospitalization when they have active or "open" tuberculosis or other conditions that make their continued employment a danger to themselves or their fellow workmen. If they have inactive pulmonary tuberculosis, or are for any other reason highly susceptible to injury by silica, they should be assigned to tasks that offer the least possible dust exposure and the maximum degree of segregation from healthy miners. All men in this group should report to the medical department after any absence from work because of illness and should have thorough re-examinations frequently (not greater than six-month intervals) to prevent the development of a significant disability before suitable measures can be initiated to maintain their health.

Regular employees who have silicosis complicated with tuberculosis or other pulmonary abnormalities should be regarded as cases of definite occupational disease, and usually should be recommended for pension or compensation if these benefits are available. Active or "open" cases of tuberculosis should be urged to accept hospitalization; men with inactive tuberculosis who are able to continue some form of employment should be selected for occupations (such as pumpman) involving little physical effort and no serious dust exposure. The men in this group should be segregated from the healthy miners so far as possible and should be re-examined after each absence because of illness as well as at frequent regular intervals to control the development of further disability.

VI—SUGGESTED ENGINEERING CONTROL PROCEDURES FOR SILICOSIS HAZARDS IN MINES

After determining the values to be assigned to the factors that comprise the degree of exposure, engineers can devise a practical plan for reducing the magnitude of a silicosis hazard. Usually this is accomplished by eliminating dust-producing operations, diluting potentially hazardous atmospheres to reasonably safe limits with large volumes of clean air, suppressing the generation of dust at the point of origin, applying protective equipment to both machines and miners, and educating the entire mine personnel in dust-control procedures.

The composition of air-borne particulate matter is best determined by analyzing several grams of sample collected from large quantities of a mine atmosphere with a suitable device such as a vacuum cleaner with a paper bag lining or an electric precipitator fitted with a small blower. A sample obtained in this manner should then be classified by sedimentation[49] or elutriation into at least three size fractions: (1) greater than 10 microns, (2) 10 to 3 microns, (3) less than 3 microns. Dust from each of these classifications, together with the unparted sample, should be subjected to a conventional chemical analysis for a determination of its constituents. If any sample discloses 10.0 per cent or more of total silica, additional tests[50,51] should be made to ascertain the proportions of this material combined with other elements or existing in the free state and to identify each of the principal mineralogical components petrographically. The size distribution may be studied after sedimentation[49] or by preparing properly dispersed samples[52] on a glass slide. Measurements are made with a microscope equipped with a filar micrometer or by microprojection[53] on a calibrated screen. A sufficient number of particles should be measured to obtain a fairly straight line when the data are plotted on Hazen's log probability paper as suggested by Drinker.[54] The common physical characteristics of silica are described in Sosman's excellent treatise.[55]

The estimation of atmospheric dust concentrations can be arrived at by a great variety of methods, none of which are exact or without objectionable features. For the collection of representative samples of dust in mine air, few instruments surpass the modified Greenburg-Smith

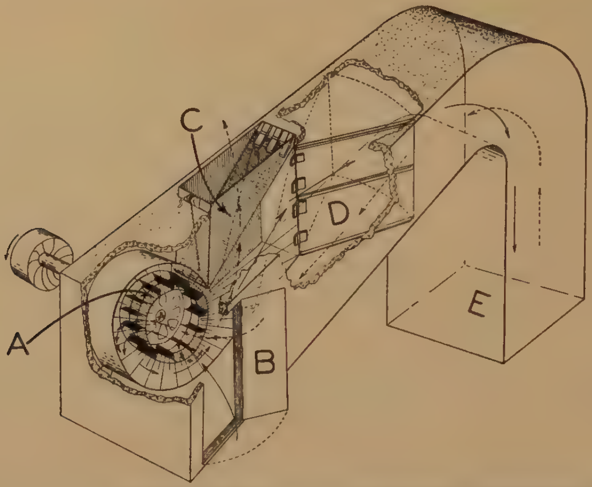


FIG. 1.—LARGE REVERSIBLE FAN FOR FORCED VENTILATION.
A, fan rotor.
B, fresh-air door.
C, exhaust door.
D, reversing door.
E, mine air shaft.

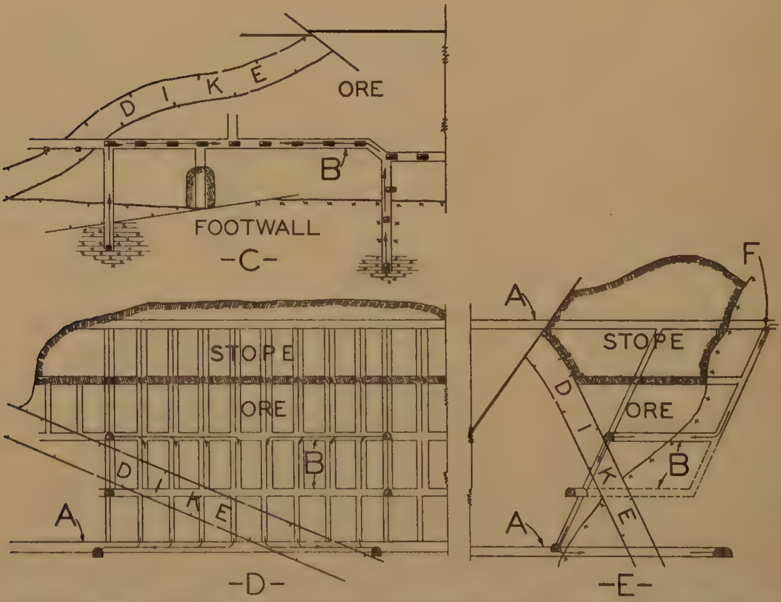


FIG. 2.—METHOD OF DUST CONTROL IN MINING AREA BY GENERAL VENTILATION.
A, main level drift.
B, sublevel drift.
C, sublevel plan.
D, footwall longitudinal.
E, cross section.
F, mesh raise door.

impinger,[56] though the Kotze konimeter[15] has been widely used in the British Dominions. If an accurate determination of the particles present in a sample of mine air is unnecessary, the simplest procedure[6] is to ascertain the weight of material suspended in a unit volume; but if a more precise measurement is desired the particles should be counted as described in U. S. Public Health Service *Bulletin* 217, modified by the addition of dark-field illumination to permit actual counting of particles down to 0.5 microns in diameter. A dark-field condenser (Zeiss' Plankton) with a low power objective (5 or 6 diameters) and a high power focusing ocular (20 or 17 diameters), when used with a good source of light (Eimer and Amend 100-watt focusing lamp with filter), provides dark-field illumination of particles within the desired size range.

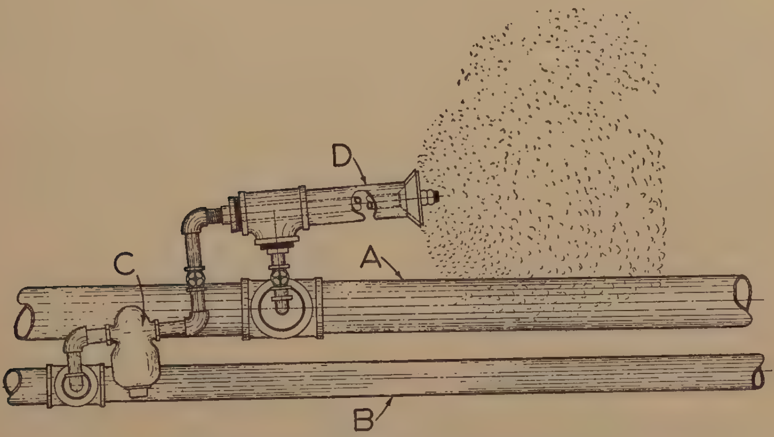


FIG. 3.—WATER BLAST FOR PRODUCING FOG.

A, air line.
B, water line.
C, water strainer.
D, water blast.

The author[4] has previously described his conception of good practice in combating dust hazards associated with mining operations and will attempt here to illustrate the principles underlying engineering supervision of these hazards by diagrams demonstrating the application of good practice to several typical operations:

Fig. 1 shows a large reversible fan suitable for controlled ventilation in an underground mine. Adequate ventilation usually requires from 300 to 800 cu. ft. of air per minute per man underground.

Fig. 2 is an example of a satisfactory plan for ventilating a mining area by means of permanent raises and crosscuts provided in the rock footwall.

Fig. 3 illustrates a water blast, which produces a fine fog that is useful in confining and saturating atmospheres contaminated with dust and powder smoke or fumes.

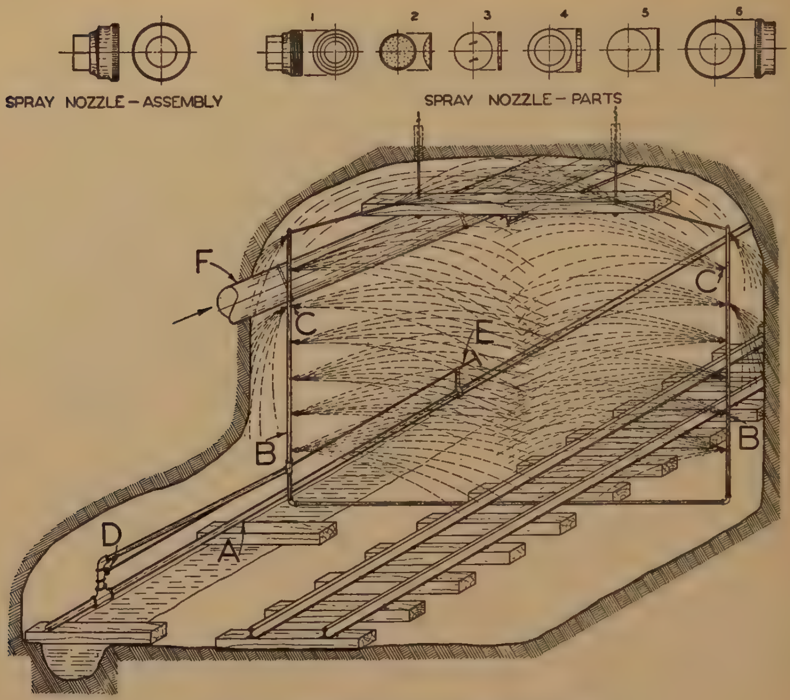


FIG. 4.—OPERATION OF WATER CURTAIN.

A, water line.
B, water curtain.
C, spray nozzles.
D, hand valve.
E, extension handle.
F, metal vent tube.

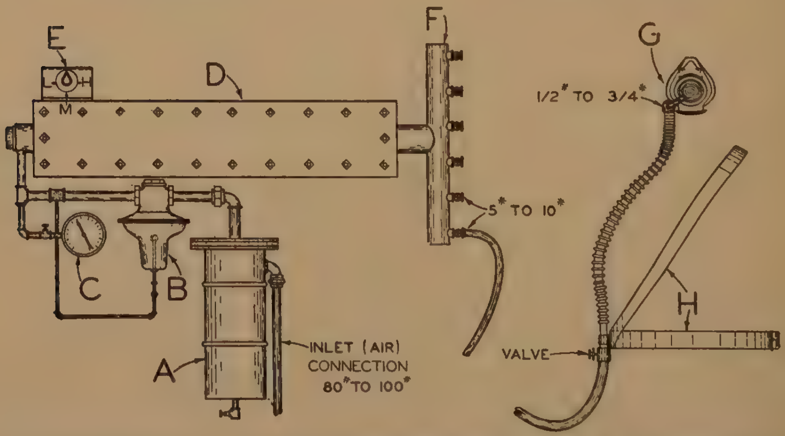


FIG. 5.—APPLICATION OF POSITIVE-PRESSURE RESPIRATORS TO UNDERGROUND AIR LINES.

A, air filter.
B, pressure reducer.
C, pressure gauge.
D, electrical space heater box.
E, heat-control switch.
F, air manifold.
G, air-line respirator.
H, worker's belt.

Fig. 4 shows a water curtain used in conditioning the contaminated air returning from a rock heading under the pressure of auxiliary forced ventilation. A number of these curtains may be installed in series to provide the required degree of dust removal.

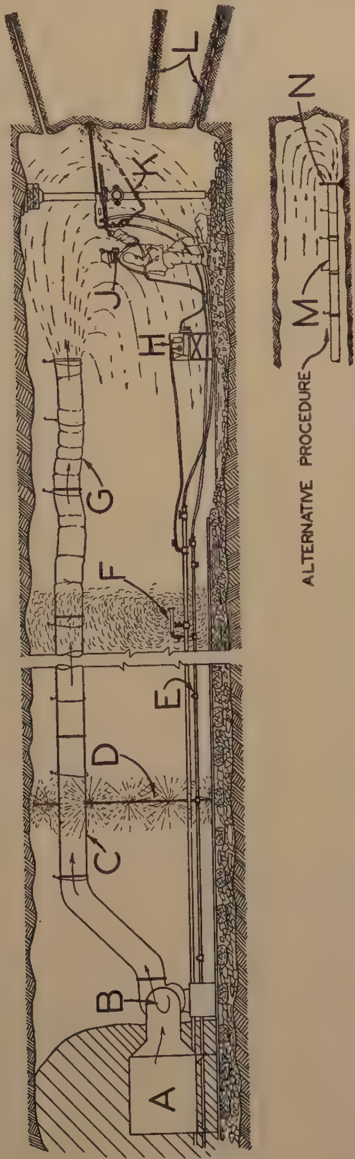


FIG. 6.—METHODS OF DUST CONTROL FOR DRIFTING OPERATIONS.

A, air filter.
B, fan.
C, metal vent tube.
D, water curtain.
E, water tees every 100 ft.
F, water blast.
G, flexible vent tube.
H, air conditioner.
J, air-line respirator.
K, water spray.
L, water in down holes.
M, metal exhaust tube.
N, baffle and pipe support.

Fig. 5 represents the application of positive-pressure air masks to underground air lines. Note that the air is filtered, reduced in pressure, heated, humidified if necessary, and delivered to the face piece through flexible tubing. Equally satisfactory are the low-pressure blowers that condition mine air and supply it to positive-pressure masks. Masks of this type furnish positive protection to miners drilling in hazardous dust concentrations.

Fig. 6 shows good practice applied to drilling in highly siliceous rock. All holes except those at the top are drilled downward to provide a water seal. The drilling is done with wet machines (preferably the African Head type) and an auxiliary spray is directed at the bit during the collaring operation. The miner wears a positive-pressure air mask. The heading is under forced ventilation from a fan situated in the fresh-air stream. The contaminated air is swept out of the heading by the proper location of the ventilating tubing and conditioned by passing through water curtains. A filter is provided for the fan if the fresh-air stream has been contaminated by considerable dust in passing through the mine.

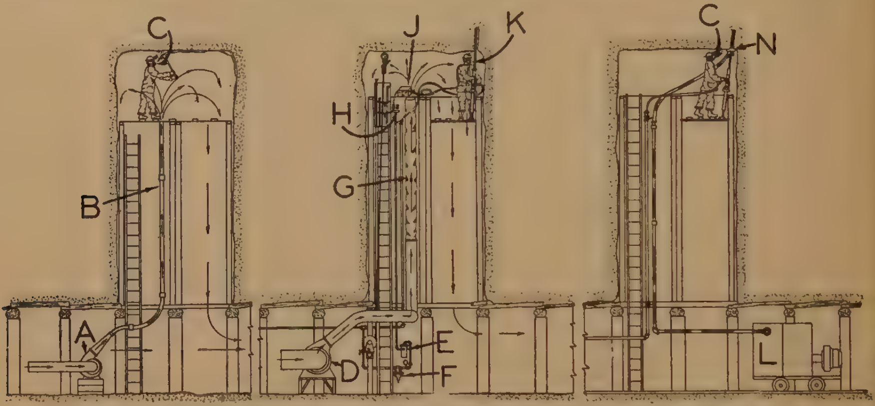


Fig. 7.—Three methods of dust control for raise operations.

A, high-pressure fan.
B, 4-in. or 5-in. pipe.
C, respirator.
D, low-pressure fan.
E, air filter.
F, pressure reducer.
G, 12-in. vent tube.
H, air manifold.
J, air heater.
K, air-line respirator.
L, dust trap.
N, suction hood.

Fig. 7 illustrates drilling in a rock raise. At the left the miner wears a filter respirator approved by the Bureau of Mines[57] and is supplied with forced ventilation through an iron pipe. In the center the miner wears a positive-pressure mask attached to the air line and is also supplied with forced ventilation through flexible tubing. Wet drilling is employed. At the right the miner is drilling a dry hole while the cuttings are removed by suction to a dust trap situated on the level below.

Fig. 8 shows miners preparing to blast a rock heading. This operation should always be done at the end of the shift. The powder is of the lowest density that will prove effective and is firmly tamped to produce the minimum of dust. All the rock surfaces adjacent to the heading are thoroughly wet down before the blast. The forced ventilation is discontinued and the tubing removed just prior to blasting. The water blast is turned on and the miners retire to fresh air. After a suitable interval the water blast is turned off. The air line to the heading is opened before miners, wearing respirators, re-enter with flexible tubing to re-establish the forced ventilation. As soon as the fan can be started

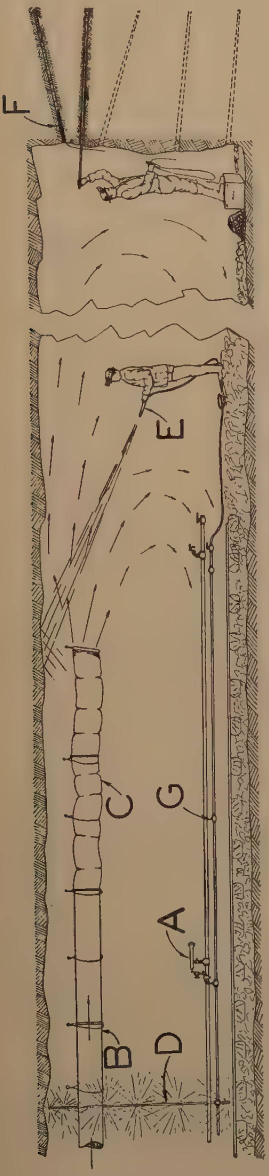


FIG. 8.—BLASTING OPERATION. BLAST AT END OF SHIFT.

Before blasting: Wet down back, walls and floor. Tamp explosives firmly. Use low-density powder. Remove vent tube. Turn on water blast.

After blasting: Fog heading with water blast. Vent air line in heading. Enter heading with respirators. Re-establish vent tube. Wet walls and broken rock.

A, water blast.
B, metal vent tube.
C, flexible vent tube.
D, water curtain.
E, water hose.
F, clay tamping.
G, water tees every 100 ft.

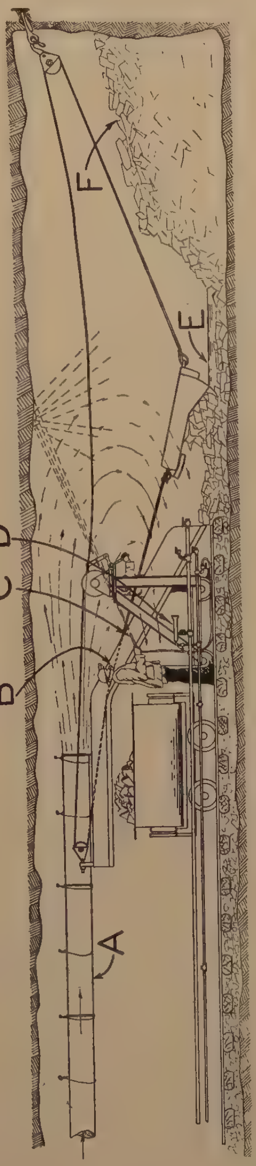


FIG. 9.—METHOD OF DUST CONTROL FOR SCRAPING OPERATIONS.

A, vent tube.
B, filter respirator.
C, water spray to wet walls and muck pile.
D, water blast.
E, water sump.
F, well wetted muck pile.

the water curtains are turned on. All rock walls are then washed down and the broken rock is thoroughly soaked.

Fig. 9 illustrates the scraping operation. Note that the miners are wearing respirators. The rock is being sprayed continually. A small sump is provided through which the scraper must travel with each load of broken rock. The heading is under forced ventilation, and the contaminated air will be conditioned by water curtains on its return through the drift.

BIBLIOGRAPHY

1. C. Badham: An Investigation Concerning Ventilation and the Sandstone Dust Present in the Air of Certain Sewer Tunnels under Construction at North Shore, and in Other Sandstone Workings. *Reports* Dir.-Gen. Pub. Health, New South Wales (1924) **52.**
2. E. P. Cowles and R. A. H. Flugge-De-Smidt: Mine Air Filtration. *Jnl.* Chem. Met. Soc. S. Africa (1934) **35,** 50.
3. D. E. Cummings: The Dust Problem in Mining Operations. *Trans.* Nat. Safety Council (1934) 281.
4. D. E. Cummings: Good Practice in Combatting Dust Hazards Associated with Mining Operations. *Trans.* A.I.M.E. (1937) **126,** 275.
5. D. E. Cummings: The Significance of Silicosis in Mining. *Min. Congress Jnl.* (Dec. 1935).
6. P. Drinker and T. Hatch: Industrial Dust. New York, 1936. McGraw-Hill Book Co.
7. J. S. Haldane: The Avoidance of Silicosis with Dry Methods of Working. *Jnl.* Chem. Met. Soc. S. Africa (1929) **30,** 54.
8. D. Harrington: Drilling and Dustiness of Metal Mine Air. U. S. Bur. Mines *R. I.* 2339 (1922).
9. D. Harrington: Dust Hazards and Their Control in Mining. U. S. Bur. Mines *I. C.* 6954 (1937).
10. S. P. Howell: Practical Mine Ventilation and Its Relation to Health and Safety; Fumes from Explosives. U. S. Bur. Mines (1937).
11. International Labour Office: Silicosis. Geneva, 1930.
12. W. R. Jones: Silicosis. *Jnl.* Chem. Met. Soc. S. Africa (1933) **34,** 99.
13. A. Mavrogordato: Contributions to the Study of Miners' Phthisis. Publications S. African Inst. Med. Res., Johannesburg, 1926.
14. R. J. Mechin, J. W. Fehnel and T. Hatch: Control of Dust in Mines (three papers.) A.I.M.E. *Tech. Pub.* 637 (1935). For the papers by Fehnel and Hatch see also *Trans.* A.I.M.E. (1937) **126.**
15. Miners Phthisis Prevention Committee: Final Report, Union of So. Africa, Johannesburg, 1919.
16. J. Moir: Recent Investigations on Dust in Mine Air. *Jnl.* Chem. Met. Soc. S. Africa (1915) **16,** 3.
17. R. R. Sayers et al.: Anthracosilicosis Among Hard Coal Miners. U. S. Pub. Health *Bull.* 221 (1935).
18. P. H. Warren: Mechanical Methods of Allaying Dust. *Proc.* Australasian Inst. Mining Met. (1921) [N. S.] **44,** 295.
19. W. Watkins-Pitchford: The Silicosis of the South African Gold Mines, and the Changes Produced in It by Legislative and Administrative Efforts. *Jnl. Ind. Hygiene* (1927) **9,** 109.

20. G. Lehmann: Die Funktion der menschlichen Nase als Staubfilter. *Arbeitsphysiol.* (1933) **7**, 167.
21. C. E. Brown: Studies in Dust Retention, III—Factors Involved in the Retention of Inhaled Dusts and Fumes by Man. *Jnl. Ind. Hygiene* (1931) **13**, 285.
22. L. U. Gardner: Pneumonokoniosis. International Clinics (1935) **2** [45], 15.
23. W. S. McCann et al.: Disability in Pulmonary Fibroses. *Jnl.* Amer. Med. Assn. (1934) **103**, 810.
24. L. U. Gardner: The Pathologic Reaction in Various Pneumonokonioses. *Jnl.* Amer. Med. Assn. (1933) **101**, 594.
25. A. C. Titus: Silica and Silicate Solubilities. *Jnl. Ind. Hygiene* (1937) **18**, 138.
26. J. McRae: The Ash of Silicotic Lungs. Publication S. African Inst. Med. Res. Johannesburg, 1913. W. E. Horter & Co.
27. A. F. Sladden: The Silica Content of Lungs. *Lancet* (1933) **2**, 123.
28. L. U. Gardner: Studies on Experimental Pneumonokoniosis, V—The Reactivation of Healing Primary Tubercles in the Lungs by the Inhalation of Quartz, Granite, and Carborundum Dusts. *Amer. Rev. Tuberc.* (1929) **20**, 833.
29. J. S. Whitney: Death Rates by Occupation. New York, 1934. Nat. Tuberc. Assn.
30. R. R. Sayers et al.: Silicosis and Tuberculosis among Miners of the Tri-State District of Oklahoma, Kansas and Missouri. U. S. Bur. Mines *Tech. Paper* 545 (1933).
31. L. Brown and H. L. Sampson: Correlation of Clinical and Roentgenological Observations in Pulmonary Tuberculosis. *Jnl. Radiology* (1934) **22**, 1.
32. J. S. Whitney: Facts and Figures about Tuberculosis. New York, 1931. Nat. Tuberc. Assn.
33. W. R. Jones: Silicotic Lungs: The Minerals They Contain. *Jnl. Hygiene* (1933) **33**, 307.
34. E. L. Collis: Industrial Pneumonoconioses with Especial Reference to Dust Phthisis. Milroy Lectures, Public Health, London (1914–15) **28**, 252.
35. E. Higgins, A. J. Lanza, F. B. Laney and G. S. Rice: Siliceous Dust in Relation to Pulmonary Disease among Miners in the Joplin District, Missouri. U. S. Bur. Mines *Bull.* 132 (1917).
36. L. U. Gardner: Studies on Experimental Pneumonokoniosis, VIII—Inhalation of Quartz Dust. *Jnl. Ind. Hygiene* (1932) **14**, 18.
37. L. U. Gardner, D. E. Cummings et al.: Unpublished Experiments of the Saranac Laboratory.
38. E. M. Chapman: Acute Silicosis. *Jnl.* Amer. Med. Assn. (1932) **98**, 1439.
39. E. L. Middleton: Silicosis in Great Britain. *Rec.* Int. Conf. Johannesburg (1930) 384.
40. L. U. Gardner and D. E. Cummings: Reaction to Fine and Medium Sized Quartz and Aluminum Oxide Particles. Silicotic Cirrhosis of the Liver. *Amer. Jnl. Path.* (1933) **9**, 751.
41. L. U. Gardner: Studies on Experimental Pneumonokoniosis, III—The Relatively Early Lesions Produced by the Inhalation of Carborundum Dust, and Their Influence on Pulmonary Tuberculosis. *Amer. Rev. Tuberc.* (1923) **7**, 344.
42. J. J. Denny, W. D. Robson and D. A. Irwin: Prevention of Silicosis by Metallic Aluminum. *Jnl.* Canad. Min. Assn. (1937) **37**, 1.
43. E. H. Kettle: Interstitial Reactions Caused by Various Industrial Dusts and Their Influence on Tuberculous Infections. *Jnl. Path. and Bact.* (1932) **35**, 400.
44. D. E. Cummings: First and Second Silicosis Symposium, Employers Mutuals, Wausau, Wisconsin, 1934 and 1935.
45. J. J. Bloomfield and L. Greenburg: Sand and Metallic Abrasive Blasting as an Industrial Health Hazard. *Jnl. Ind. Hygiene* (1933) **15**, 184.

46. L. U. Gardner: Pathology of So-called Acute Silicosis. *Amer. Jnl. Pub. Health* (1933) **23,** 1240.

47. H. L. Sampson: Second Silicosis Symposium, Employers Mutuals, Wausau, Wisconsin, 1935.

48. H. K. Pancoast et al.: Roentgenological Appearances in Silicosis and the Underlying Pathological Lesions. U. S. Pub. Health *Reprint* 1696 (1935).

49. D. E. Cummings: Studies on Experimental Pneumonokoniosis, IV—The Separation of Particulate Matter Smaller than Screen Size into Graded Fractions. *Jnl. Ind. Hygiene* (1929) **11,** 245.

50. A. Knopf: The Quantitative Determination of Quartz (Free Silica) in Dusts. U. S. Pub. Health *Repts.* (1933) **48,** 183.

51. H. L. Ross and F. W. Sehl: Determination of Free Silica, Modified Petrographic Immersion Method. *Ind. Eng. Chem.*, Anal. Ed. (1935) **7,** 32.

52. H. Green: A Photographic Method for the Determination of Particle Size of Paint and Rubber Pigments. *Jnl.* Frank. Inst. (1921) **192,** 637.

53. E. J. Dunn: Microscopic Measurements for the Determination of Particle Size of Pigments and Powders. *Ind. Eng. Chem.* Anal. Ed. (1930) **2,** 59.

54. P. Drinker: The Size Frequency and Identification of Certain Phagocytosed Dusts. *Jnl. Ind. Hygiene* (1925) **7,** 305.

55. R. B. Sosman: The Properties of Silica. New York, 1921. Chemical Catalog Co.

56. J. J. Bloomfield and J. M. Dalla Valle: The Determination and Control of Industrial Dust. U. S. Pub. Health *Bull.* 217 (1935).

57. C. E. Brown and W. Yant: Procedure for Testing Filter Type Dust, Fume and Mist Respirators for Permissibility. Schedule 21, U. S. Bur. Mines (1934).

INDEX

(Note: In this index the names of authors of papers and discussions and of men referred to are printed in small capitals, and the titles of papers in *italics*.)

A

Accidents: metal mines: compensation: Anaconda Copper Mining Co. at Butte, 244
Aerial tramway: Ymir Yankee Girl Gold Mines Ltd., 99
Africa: Belgian Congo. *See* Belgian Congo.
Agua Fria mine, Honduras: airplane transportation of equipment and supplies, 225
Air conditioning (*see also* Air Cooling):
  Butte mines: procedure, 271
  Magma mine: cost, 263
    effect, 265
    procedure, 253
Air cooling (*see also* Air Conditioning):
  deep gold mines: heat loads: calculation, 282
    heat loads: measurements, 283
    dry-bulb temperature real criterion for heat flow from rock to air, 287
    on the Rand, 276
  deep mines: brief bibliography, 287
Airplane transportation of freight: Agua Fria mine, Honduras, 225
  cost: Agua Fria mine, Honduras, 226
    aviation company, 227
    Bulolo Gold Dredging Co., 228
    Canadian bush, 228
    single operator, 226
    United States, 229
Alabama: prospecting, mining and washing brown iron ores, 107
Alenius, E. M. J.: *Discussion on Preliminary Stripping of the Morenci Open Pit, Arizona*, 61
Anaconda Copper Mining Co.: safety and health efforts at Butte, 241
  ventilation and dust prevention: Anaconda Copper Mining Co. at Butte, 268
Anchor tower, 23
Anderson, A. E., Riddell, J. M. and Holt, G. J.: *Recent Improvements in Mining Practice on the Mesabi Range*, 9

B

Belgian Congo: Katanga: copper deposits: types, 27
Berggren, R. C.: *Hot-milling of Rock-drill Bits*, 211
Bjorge, G. N.: *Discussion on Sand Filling at the Homestake Mine*, 161, 162
Blasting: electric: Climax, Colo., 165
    delay caps, 166, 174, 175
    formulas for calculation of requirements, 168
    graded series effect, 173
    instantaneous caps, 166, 174, 177
    loop-back system, 165
    recommendations, 175
    stray currents, 177
    three-wire system, 165, 170
    two-wire system, 174, 175, 176
Boardman, J. L.: *Safety and Health Efforts of the Anaconda Company at Butte*, 241
Bowen, R. A.: *Sublevel Caving, Large-pillar Method, at the Montreal Mine*, 119
Bulolo Gold Dredging Co.: cost of airplane transportation of freight, 228

C

Canada: airplane transportation: cost, 228
Cananea Consolidated Copper Co.: Colorada mine: controlling subsidence of large inverted cone of barren rock lying above ore body, 181
Carrene: refrigerant, 276
Carrier, W. H.: *Air Cooling in the Gold Mines on the Rand*, 276
Carrier Corporation: air cooling in the gold mines on the Rand, 276
Catron, W. and Cooledge, C. U.: *Controlling Subsidence of a Large Inverted Cone of Barren Rock Lying Above the Ore Body, Colorada Mine, Cananea Consolidated Copper Company*, 181
Climax Molybdenum Co.: big blasting, 165
Colorada mine: controlling subsidence of a large inverted cone of barren rock lying above ore body, 181
Compensation. *See* Accidents.
Conveyor belts: iron mining: Mesabi Range, 15
  versus rail haulage, 26
Copper deposits: Belgian Congo: types, 27
  Kipushi, Katanga, Belgian Congo, 27
Copper mining: open pit: preliminary stripping: Morenci, Arizona, 47
  top slicing: Globe-Miami district, Arizona, 44
    Kipushi, Katanga, Belgian Congo, 27
    Morenci, Arizona, 42
Cromwell, R. H.: *Discussion on Sand Filling at the Homestake Mine*, 162

- Cooledge, C. U. and Catron, W.: *Controlling Subsidence of a Large Inverted Cone of Barren Rock Lying Above the Ore Body, Colorada Mine, Cananea Consolidated Copper Company*, 181
- Cuba. *See* Matahambre.
- Cummings, D. E.: *Suggestions for the Control of Silicosis in Mining*, 288

D

- Denny, E. H.: *Hazards from Gases in Metal Mines and Protections against Them*, 230
- Dick, G. M. and Haight, H. V.: *Discussion on Some Outstanding Mine-hoisting Equipment*, 203
- Diesel engines: copper mining: Morenci, Arizona, 61
  - use on Mesabi Range, 9
- Dragline buckets: without bottoms: advantages, 25
- Drill bits (*see also* Rock Drills):
  - design: cooperation between operators and machinery manufacturers, 79
  - detachable bits: results in lead-zinc ore, 67
  - forged: results in lead-zinc ore, 67
  - hot-milling: advantages, 218
    - approaching cut, 214
    - eastern customs shop, 217
    - operation at western mine, 215
    - process, 211
    - receding cut, 213
    - uniformity in presentation of data, 80
  - Picher bit, 66, 79
  - reconditioning: grinding vs. hot-milling, 218
    - hot-milling: process, 211
- Drill steel (*see also* Rock Drills):
  - durability: lead-zinc ore: Tri-State district, 78
  - gauge: 1/16 inch: advantages at Homestake, 220
    - Homestake, 220
  - Swedish: results in lead-zinc ore, 68
- Drilling and blasting: lead-zinc ore: Tri-State district, 64
- Dust: metal mines: characteristics at Butte, Mont.: determining, 273
    - concentration at Butte, Mont., 275
    - control: blasting operations, 305
      - drifting operations, 303
      - positive-pressure air masks, 303
      - raise operations, 304
      - scraping operations, 305
      - ventilation, 300
      - water blast, 301
      - water curtain, 302
    - prevention at Butte, Mont., 272

E

- Eagle-Picher Mining and Smelting Co., Picher, Oklahoma, 62
- Eaton, L.: *Discussions: on A New Method of Top Slicing at Kipushi, Katanga, Belgian Congo*, 45
- Eaton, L.: *Discussions: on Sand Filling at the Homestake Mine*, 162

F

- Flotation: gold: Ymir Yankee Girl Gold Mines Ltd., 104
- Foraker, C. B.: *Ventilation and Air Conditioning of the Magma Mine*, 253

G

- Gas: mine. *See* Mine Gas.
- Globe-Miami district, Arizona: top slicing, 44
- Gogebic Range: Montreal mine. *See* Montreal Mine.
  - shaft sinking, 128
- Gold milling: Ymir Yankee Girl Gold Mines Ltd., 102
- Gold mines: air cooling on the Rand, 276
  - geology: Ymir-Salmo district, 82, 84
  - Ymir Yankee Girl. *See* Ymir.
- Gold mining: costs: Ymir Yankee Girl Gold Mines Ltd., 96
  - sand filling: Homestake mine, 128
  - Ymir Yankee Girl Gold Mines Ltd., 96

H

- Haight, C. M.: *Discussion on Sand Filling at the Homestake Mine*, 160
- Haight, H. V. and Dick, G. M.: *Discussion on Some Outstanding Mine-hoisting Equipment*, 203
- Health: Anaconda Copper Mining Co.: efforts at Butte, 241
- Healy, R. L.: *Discussion on Shaft Sinking on the Gogebic Iron Range*, 145
- Hoists: mine: automatic: Inspiration Copper Co., 190
    - automatic: Mufulira Copper Mines, 190
    - balancing rope weight, 193
    - brakes for drum, 197
    - clutches, 194, 203, 205
    - drums: types, 191
    - handling men and ore, 191
    - pulleys: Bollen, 198
      - Karlik, 199
      - Koepe: mechanics underlying, 202
        - operation, 200
      - Ohnesorge: mechanics underlying, 202
        - operation, 200
    - types, 189
    - two-speed, 191
- Hoisting: mine: brief bibliography, 202
    - choice of hoist: factors governing, 189, 201, 204
    - equipment: review of some outstanding, 188
- Holt, G. J., Anderson, A. E. and Riddell, J. M.: *Recent Improvements in Mining Practice on the Mesabi Range*, 9
- Homestake mine: sand filling, 146
- Homestake Mining Co.: drill-steel gauge, 220

Honduras: airplane transportation. *See* Agua Fria.
Hot-milling. *See* Drill Bits.

I

Indian lands: leases for mining, Tri-State district, 62
Insurance: group: Anaconda Copper Mining Co. at Butte, 250
Iron mining: Mesabi Range. *See* Mesabi Range.
  shaft sinking: Gogebic Range, 128
    through granite, 128
  sublevel caving: Montreal mine: *See* Montreal Mine.
  open-pit, 13
  trucks and tractors in use, 10
Iron ores: brown: Alabama: districts mined, 111
    churn-drilling: Alabama, 110
    mining: Alabama, 111
    price: Alabama, 118
    production: Alabama, 1928–1936, 108
      United States, 1928–1936, 108
    prospecting: Alabama, 107
    sampling: Alabama, 110
    tailings: method of treating needed, 117
    washing: Alabama, 113
  Mesabi Range: characteristics, 26

J

Jackson, C. F.: *Discussion on Sand Filling at the Homestake Mine*, 161
Japan: rock drills: tests on 67 drills, 206
Johnston, V. D.: *Discussion on Shaft Sinking on the Gogebic Iron Range*, 145

K

Katanga. *See* Belgian Congo.
Kipushi, Katanga, Belgian Congo: copper deposits, 27
Knoll, W. A.: *Discussions: on Shaft Sinking on the Gogebic Range*, 145
  on Sublevel Caving, Large-pillar Method, at the Montreal Mine, 127
Knoll, W. A. and Sullivan, J. C.: *Shaft Sinking on the Gogebic Iron Range*, 128

L

Lawson, W. C.: *Preliminary Stripping of the Morenci Open Pit, Arizona*, 47
Lead-zinc mining (*see also* Tri-State District and Zinc-lead):
  Tri-State district: improvements in practice, 62
Leidich, H. A.: *Discussion on A New Method of Top Slicing at Kipushi, Katanga, Belgian Congo*, 41
Lindell, K. V.: *Discussion on Recent Improvements in the Mining Practice of the Tri-State District*, 80
Lorain, S. H.: *Discussion on Gases in Metal Mines*, 239

M

Magma Copper Co.: ventilation and air conditioning of the Magma mine, 253
Magma mine: ventilation and air conditioning, 253
Matahambre, Cuba: sand filling, 162
McIntyre, J. T.: *Discussion on Air Cooling in the Gold Mines on the Rand*, 287
McNicholas, F. S.: *Progress of "Big Blasting" at Climax*, 165; *discussion*, 178
Mesabi Range: conveyor belts, 15
  mining practice: improvements, 9
  open-pit mining, 13
  ores: physical characteristics, 26
  trucks and tractors, 10
Mine gas: metal mines: asphyxiation from: typical examples, 230
    detection, 235
    exhaust from gasoline engine, 234
    from strata, 231
    hazards and protection against, 230
    hydrogen sulphide, 235
    methane, 235
    oxygen-deficient air, 235
    produced by blasting, 232
    protection against, 235
Mining methods: anchor tower, 23
  open-pit: Mesabi Range, 13
    preliminary stripping: Morenci, Arizona, 47
  sand filling: Homestake mine, 128
  sublevel caving: large-pillar method: Montreal mine, 119
  surface: classification, 179
  top slicing: Globe-Miami district, Arizona, 44
    Morenci, Arizona, 42
    new method at Kipushi, Belgian Congo, 27
Montreal mine: ore: occurrence and characteristics, 119
  sublevel caving: large-pillar method, 119
Morenci, Arizona: open-pit mining: preliminary stripping, 47
  top slicing, 42
Morgan, C.: *Prospecting, Mining and Washing the Brown Iron Ores of Alabama*, 107
Morrell, L. G. and Wright, L. B.: *Ymir Yankee Girl Gold Mines Ltd.*, 81
Mosier, Mc H.: *Discussion on Improvements in Mining Practice on the Mesabi Range*, 26

N

Newport iron mine: shaft sinking, 128
Nicolson, C. W.: *Recent Improvements in the Mining Practice of the Tri-State District*, 62; *discussion*, 78, 79, 80
Nordberg, B. V. E.: *Some Outstanding Mine-hoisting Equipment*, 188; *discussion*, 204

O

Open-pit mining: copper: Morenci, Arizona, 47

P

Paynter tester: use in testing rock drills, 206
Phelps Dodge Corporation: preliminary stripping of the Morenci open pit, Arizona, 47
Pickands Mather & Co.: shaft sinking on the Gogebic iron range, 128
Prince Leopold mine, Kipushi, Katanga, Belgian Congo: top slicing, 27

R

Read, T. T.: *Discussions: on Improvements in Mining Practice on the Mesabi Range*, 25
    on Recent Improvements in the Mining Practice of the Tri-State District, 79, 80
Rice, G. S.: *Discussions: on Improvements in Mining Practice on the Mesabi Range*, 25
    on Sand Filling at the Homestake Mine, 161
    on Shaft Sinking on the Gogebic Iron Range, 145
Richardson, A. S.: *Ventilation and Dust Prevention in the Butte Mines*, 268
Riddell, J. M.: *Discussions: on Improvements in Mining Practice on the Mesabi Range*, 25, 26
    on Shaft Sinking on the Gogebic Iron Range, 145
Riddell, J. M., Anderson, A. E. and Holt, G. J.: *Recent Improvements in Mining Practice on the Mesabi Range*, 9
Robinson Deep mine: air cooling, 276
Rock drills (*see also* Drill Bits, Drill Steel):
    design: Suzuki's recommendations, 210
    discussion of designs of valves, cylinders, piston hammers and rotation devices (abstract), 206
    experimental study (Suzuki), 206
    tests on 67 American, Japanese and European drills, 206
Ross, A. J. M.: *Sand Filling at the Homestake Mine*, 146; *discussion*, 163
Ruhloff, F. C.: *Discussion on Improvements in Mining Practice on the Mesabi Range*, 25

S

Safety: Anaconda Copper Mining Co.: efforts at Butte, 241
    metal mines: bonuses: Anaconda Copper Mining Co., 247
Sand filling: advantages, 160, 163
    cost: Homestake mine, 159, 162
    disadvantages, 160
    drainage, 158, 161
    Homestake mine, 165
    Matahambre, Cuba, 162
    pipe size: relation to amount deposited, 162
    pipe wear, 158, 161
    screen analyses of sand, 163
    stope lacing, 163
Saranac Laboratory: suggestions for control of silicosis in mining, 288
Scraper loading: lead-zinc ore: Tri-State district, 64
Shaft sinking: in granite: vertical holes, 128
    without V-cut, 128
Shay, F. B.: *Air Transport at Agua Fria*, 225
Sherman, G.: *Discussions: on Progress of "Big Blasting" at Climax*, 178
    on Recent Improvements in the Mining Practice of the Tri-State District, 78
    on Sand Filling at the Homestake Mine, 161, 162
Silicosis (*see also* Dust, Health, etc.):
    control in mines: bibliography, 306
        objectives, 295
        positive-pressure air masks, 303
        suggestions, 288
        ventilation, 300
        water blast, 301
        water curtain, 302
    engineering control: suggested procedure, 299
    harmful exposure: degree, 292
    hazard at mine: determining, 296
    individual susceptibility: factors comprising degree, 288
    medical control at mines: suggested procedure, 296
Smith, F. J.: *Discussion on Shaft Sinking on the Gogebic Iron Range*, 145
Spruce mine: Mesabi Range: conveyor belts in mining, 18
Stokes, W. E. D. Jr.: *Discussion on Air Transport*, 228
Subsidence: inverted cone of barren rock above ore body: controlling, 181
Sullivan, J. C. and Knoll, W. A.: *Shaft Sinking on the Gogebic Iron Range*, 128
Suzuki, T.: *An Experimental Study of the Rock Drill (Digest by B. F. Tillson)*, 206

T

Tillson, B. F.: *Digest of An Experimental Study of the Rock Drill (Suzuki)*, 206
    Discussions: on Improvements in Mining Practice on the Mesabi Range, 25, 26
    on Recent Improvements in the Mining Practice of the Tri-State District, 79, 80
    on Some Outstanding Mine-hoisting Equipment, 202
Tractors: iron mining: Mesabi Range, 9
    Mesabi Range: iron mining, 9
Tri-State district: drilling and blasting practice, 64
    drill steel: 64, 78
    headframes, 77
    lead-zinc ore deposits: metal content, 63
        types, 63
    mining practice: drifting and stoping, 68
        drilling and blasting, 64
        exploration, 64
        improvements, 62
    safety and welfare, 77
    scraper loading, 69
    tramming and hauling, 75

Trucks: bottom-dump: iron mining, 12
  end-dump: iron mining, 11
  iron mining: Mesabi Range, 9
  Mesabi Range: iron mining, 9
  open pit mining: Morenci, Arizona, 58, 61
  side-dump: iron mining, 11

U

Union Minière du Haut Katanga: top slicing at Kipushi, Katanga, Belgian Congo, 27

V

Van Esbroeck, G. and Van Weyenbergh, M.: *A New Method of Top Slicing at Kipushi, Katanga, Belgian Congo*, 27
Van Evera, J. W.: *Discussion on Improvements in Mining Practice on the Mesabi Range*, 25, 26
Van Weyenbergh, M. and Van Esbroeck, G.: *A New Method of Top Slicing at Kipushi, Katanga, Belgian Congo*, 27
Ventilation: Butte mines: procedure, 268
  copper mine: top slicing: Kipushi, Katanga, Belgian Congo, 37, 45
  Magma mine: cost, 258
    procedure, 253

W

Walker, H. A.: *Recent Trend in Drill-steel Gauge at Homestake*, 220
Weysser, J. L. G.: *Discussion on Shaft Sinking on the Gogebic Iron Range*, 145
Witwatersrand: air cooling deep mines, 276
Workmen's compensation. *See* Accidents.
Wright, L. B. and Morrell, L. G.: *Ymir Yankee Girl Gold Mines Ltd.*, 81

Y

Ymir Yankee Girl Gold Mines Ltd.: aerial tramway, 99
  development, 97
  drainage, 98
  geology of district, 82, 84
  history, 81
  milling, 102
  mining costs, 100
  mining methods, 96
  production, 81
  sampling and estimation, 96
  stoping, 97
  underground transportation, 98
  ventilation, 99

Z

Zinc-lead mining: drilling and blasting practice: Tri-State district, 64
  Tri-State district: improvements in practice, 62

3 8198 309 333 183
THE UNIVERSITY OF ILLINOIS AT CHICAGO

THIS BOOK IS FOR USE
ONLY IN THE LIBRARY
IT DOES NOT CIRCULATE